METAL MINING
MILLING METHODS

1943

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FOREWORD

In a sense this is a two-in-one volume of TRANSACTIONS: it contains the cream of papers on crushing, ore dressing, and concentrating accumulated since the issuance of the Milling Methods volume of 1939 and of papers on Metal Mining accumulated since the last volume on that subject, which was published in 1940. All of the papers in the volume have appeared in MINING TECHNOLOGY. Instead of publishing a volume on Milling in 1943 and one on Mining in 1944, the Directors chose to combine the two. The community of interest between those engaged in digging ores and in dressing them is close in several respects. The operations usually are conducted in one community and the local management usually is the same. A more important tie is that the efficiency of milling frequently depends upon the way mining operations are conducted. For the optimum over-all result there should be close cooperation and coordination between these two departments of an enterprise.

The Directors of the Institute recognize that technologic change in mining and milling has not been as significant as in some other of the Institute's various fields. However, they feel that, with respect to mining, at least, coverage has not been as complete as desirable; and it is hoped that this will be corrected by the submission of more papers describing progress in the art. So-called "practical" or operating papers are particularly welcome.

The various committees under whose sponsorship the volume is published are shown on pages 10 and 11. To these committees, and especially to the various chairmen, and to the authors of the articles, the industry, the profession and the Institute are indebted.

A. B. PARSONS, *Secretary.*

NEW YORK, N. Y.
September 1, 1943.

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Mining and Transportation Practice in Minnesota Iron Mines

By Grover J. Holt,* Member A.I.M.E.

(Duluth Meeting, August 1941, and New York Meeting, February 1942)

A detailed description of the many variations in iron mining and transportation practice in Minnesota would require much space. Since a fairly detailed description of the practices then in use was published in the guidebook issued in connection with the meeting of the Institute on the iron ranges in 1920, and subsequent developments have been described by E. E. Hunner,[1] M. H. Barber,[2] and A. E. Anderson, J. M. Riddell and G. J. Holt,[3] it will not be necessary here to do more than review the highlights of current practice. Details on any points of interest can be obtained through correspondence with the operators involved.†

The iron ores of Minnesota, which have been formed through enrichment of the iron formation in which they occur, largely by the removal of silica, are usually covered by a blanket of glacial drift known locally as "surface." Surface material, including sand, clay, hardpan and boulders, makes up the greater portion of the stripping in open-pit operations. In some instances, stripping also includes taconite and "paint rock," directly below the surface material and overlying the ore. Taconite is the iron formation, a ferruginous chert with interbedded slate layers. "Paint rock" is a local term for oxidized slate layers interbedded in the iron formation, high in phosphorus, alumina and moisture.

The iron ores fall into two classes: (1) direct shipping (merchantable ores or ores requiring no treatment except possibly crushing or screening to make an acceptable sized product), and (2) ores requiring beneficiation (such as sintering, drying, washing, jigging, high-density treatment, or magnetic separation in the case of magnetite).

The Mesabi Range produces chiefly soft Bessemer and non-Bessemer ores. The Cuyuna Range produces mainly low-manganese brown ore and high-manganese black ores, with some iron ore. The Vermilion Range yields mainly Bessemer and non-Bessemer iron ore, with some hard or lump ore that finds use in the open hearth. Open-pit mining is limited to the Mesabi and Cuyuna Ranges but underground mining is done on all three. The choice of mining methods is determined by the relative economics of overburden-removal and shaft-mining costs.

Stripping Practice

Common practice has been to remove the bulk of the overburden during the winter season and to mine during the summer. In the present emergency many operators are forced to forego the economies attained by year-round utilization of equipment and to equip for, and carry on, stripping and mining simultaneously. In the normal years or lean years to come, most properties probably will be overequipped and high depreciation charges will have to be absorbed.

Manuscript received at the office of the Institute July 31, 1941. Issued in Mining Technology, March 1942.

* Butler Brothers, St. Paul, Minn.

[1] References are at the end of the paper.

† To obtain information for this composite picture of open-pit and underground operations, questionnaires were sent out, to which 32 open pits and 7 underground mines responded. The writer wishes to acknowledge the cooperation of the Oliver Iron Mining Co., Pickands Mather and Co., M. A. Hanna Co., Cleveland Cliffs Co., Republic Steel Corporation, Inland Steel Co., Snyder Mining Co., Wheeling Steel Co., North Range Mining Co., and Butler Brothers in the preparation of this material.

In calculating the yardage of stripping to be removed from the pit area, all classes of material are reduced to an equivalent yardage basis by applying factors as follows:

Class of Material	Times Actual Yardage
Surface material	1
Paint rock	1½ to 1¾
Taconite	2 to 2½

The actual yardage of each class of material from engineers' estimates are multiplied by these factors to determine equivalent yardage. The sum of these products is the total equivalent stripping, since it is assumed that a yard of paint rock will cost from 1½ to 1¾ times, and taconite from 2 to 3 times, as much to move as a yard of surface material. The companies use various factors, generally within these ranges.

The maximum ratio of equivalent stripping to tons of ore in past years was considered to be about 3 yd. of stripping per ton of ore. Any ore body having a greater ratio than that was classed as more of an underground than an open-pit possibility. Today this stripping ratio has increased to as much as 5 to 1 in some places. Of the 32 pits reported, actual stripping depths ran from 32 ft. as a minimum to 135 ft. as a maximum.

Drilling and Blasting.—Surface material seldom is drilled and blasted prior to loading in summer, but frequently requires it in winter; paint rock and taconite usually require "shaking" prior to loading. Drills for this work are typically of the churn type, used with or without casing. Common practice a few years ago, but less frequently used today in surface materials, is gopher-holing, a horizontal hole opened by light charges of dynamite, and cleaned out by means of long-handled shovels or spoons. This method can be used only in well consolidated materials. Black powder has given way to the less dangerous, more easily handled dynamite.

Loading.—Shovels may be powered with gas, Diesel, or electric motors. In the smaller pits, and for cleaning the top of ore, gas and Diesel shovels of 1½ to 3 cu. yd. capacity predominate because of greater mobility. In larger pits the electric machine is by far the favorite, in 3 to 6½-yd. sizes. Several open pits were stripped years ago and have had no recent stripping, so any compilation of the type of equipment used would be misleading as to present practice, but the statistics listed under ore mining will serve to indicate present trends.

Haulage.—Until about 1937, most of the haulage was done by locomotives and cars. Since then the use of trucks has increased rapidly. No actual figures are available but it seems safe to say that the bulk of the stripping handled in 1941 was transported by truck. These same trucks have seen service in the various stripping operations and are now being used in hauling ore.

The trucks used in the haulage of stripping thus far have been limited in capacity to 15 tons, and run from this size down to the smaller, standard, dump truck that is used by road contractors.

Of the 32 pits reported, 23 have used steam locomotives and 9 have used trucks on the haulage of stripping. Of the 23 open pits utilizing locomotive and car haulage on stripping, 9 have changed over to truck haulage, and one to caterpillar-wagon haulage on ore. Today only two open pits have standard-gauge electrified railroad haulage (one of which was unreported). Most stripping haulage by locomotives on the Ranges employs either steam or electric locomotives weighing from 60 to 65 tons. The capacities of dump cars, which are utilized largely for stripping, are reported as 15, 16, 20 and 30 cu. yd. The 30-cu-yd. size predominates, followed by the 20-cu.-yd. cars.

Tracks.—Rail in use varies from 80 to 120 lb. per yard; tracks are laid and transported usually in sections. For short moves track shifters are sometimes used, and temporary stripping tracks are seldom ballasted for this service.

Stripping Dumps.—Stripping removed from an ore body is dumped beyond the limit of possible occurrence of ore. Dumps for locomotive and car haulage may be a mile or more from the open-pit limits, but truck-haulage dumps are placed as near as possible to the open-pit limits usually only a few thousands of feet. The reason for the longer locomotive haul lies in the necessity for limiting grades, which, combined with desired dump heights, requires the additional distance. Table 1 shows the distances of stripping haul.

TABLE 1.—*Distances Stripping Is Hauled*

Distances	Number of Pits	
	Locomotives and Cars	Trucks
Under 3000 ft.		4
3000–4000 ft.		4
Under 1 mile	3	
1 mile or over		1
1–2 miles	11	
2–3 miles	6	
3 miles or over	2	

For locomotive and car haulage the common practice is to build wooden trestles in order to start the dump at the required height. This has been necessary because of the flatness of the terrain on the iron ranges. The height of dump is increased as filling progresses by raising the fill at each move of the track. Some dumps have been started from ground levels and raised by jacking and filling. The track usually is shifted by a track shifter, locomotive crane, or a boom-type steam-operated track shifter; today there is very little shifting of tracks by hand. Truck and caterpillar haulage requires no trestle and the dump may be raised on an 8 per cent grade as desired. Bulldozers are always essential in dump maintenance, to eliminate mounds at the crest of the dump and to provide roadways across it.

Stripping Roads.—For truck haulage of stripping, since the roads are temporary, grading commonly is done by means of bulldozer or patrol grader. The locus of the stripping approach is changed so frequently that any expenditure for a finished roadway would be almost entirely lost after one year of operation. If a section of stripping road becomes soft in wet weather a light surface of gravel or mill rejects is added occasionally. On the stripping roads in the pits, bulldozers are used at the shovel to level out spillage and to cut run-around roads but mainly for clean-up work. The top of ore is an excellent road material in itself. Roads are inspected by patrolmen to eliminate sharp rocks, wood, and stray steel, which are extremely destructive to tires.

Frost Prevention.—During the winter, stripping material freezes in the bodies of the cars and trucks, so salting stations near stripping tracks or roads provide hot brine solutions. The hot brine is sprayed inside the bodies of cars and trucks. Even in spite of this precaution a great deal of hand-cleaning of bodies is necessary during subzero weather.

Open-pit Mining of Ore

Drilling.—After stripping, the ore is drilled, usually by churn drills equipped with bailers, to the depth required to provide the proper height of bank for shovel loading. Often the depth of hole is governed by ore-grading requirements, because of the various classes of ore that occur in almost every pit. These holes are 6 to 8 in. in diameter and casing is used only where necessary during the drilling period. The average depths of holes are as follows:

Feet	15–19	20–24	25–29	30–34	35–40
Number of pits	4	9	5	5	4

Recently a new type of auger drill has been adopted for drilling a 6-in. horizontal hole, and this has proved practical. At the 32 open pits reported only two jackhammer drills are listed; 42 churn drills and 6 horizontal auger drills are used.

Blasting.—Black powder has been largely displaced by dynamite, of several grades.

The tons of ore broken per pound of explosive, ready for shovel loading, vary as follows:

Tons broken	2–4	4–6	6–8	8–10	10–12	12–14	14–16
Number of pits	3	9	4	3	2	2	3

As a rule, ores from the Mesabi and Cuyuna Ranges require only bank blasting in preparation for loading. In some places, however, mud capping must be used on the larger lumps so that they can be handled by the loading or haulage equipment.

Table 2.—*Electric and Steam Shovels and Haulage Equipment*

Method of Haulage	Haulage Equipment Electric Shovels, Cu. Yd., and Number												
	6½	6	5½	5	4½	4	3¼	3	2½	2¼	2	1¾	1½
Locomotive and cars	3	1	2	9		17	1	4	2		1		1
Trucks					1	8	4	4	4	2	1	1	
Caterpillars and wagons													

Method of Haulage	Steam Shovels, Cu. Yd., and Number				
	8	4	2	1¾	½
Locomotive and cars	1	2	3	1	1
Trucks			2		
Caterpillars and wagons					

Loading.—Shovels vary in size, depending on the type of haulage used and the tonnage of output desired. Smaller pits utilize shovels of from 1½ to 3 cu. yd., and the larger pits from 4 to 6½ cu. yd. capacity. One open pit on the Mesabi Range utilizes two tower excavators[3] with 3-cu.-yd. crescent open-bottom scrapers for the loading of ore into raises in connection with conveyor-belt haulage, this being the only type of loading tool other than standard shovels used in ore production. Table 2 shows the sizes of shovels used, together with the type of haulage equipment into which the ore is loaded. While this table does not include unreported open pits, it gives an excellent cross section of the larger operations on the Ranges. It is evident that all shovels larger than 4½ cu. yd. are confined to pits still using locomotive and car haulage. All of these larger machines are electric except one, which shows the effect of obsolescence on the old large steamers. The favorite for locomotive and car service is the 4-cu.-yd. electric and this is preferred also for truck loading (Table 2). Of the steam shovels remaining, all but two are used with locomotive and car service.

Gasoline and Diesel shovels (not included in table) in sizes of 1 to 1¼ yd. are used only in truck and caterpillar-wagon service. Shovels of this type from 1½ to 2 yd. are frequently used for ore stockpile loading but seldom for open-pit work.

Table 3.—*Ore Haulage to Yards or Plant*

Steam locomotives and cars	11
Electric locomotives and cars	2
Electric locomotives and cars with incline hoist	1
Steam and electric locomotive and shaft	1
All trucks	7
Trucks and steam locomotive	3
Trucks and conveyor belt	4
Trucks, conveyor belt and steam locomotive	2
Caterpillar wagon and conveyor belt	1

Haulage.—The greatest change in the iron-mining industry has been in ore haulage. Table 3 shows the various methods of hauling ore either to the railroad gathering yards or to the washing plant of the open pits. The maximum and average grades, together with the lengths of haul for steam and electric locomotives, are included in

Table 4. Table 4 seems to indicate that two pits report standard-gauge electric locomotive haulage, but actually only one pit is represented; two adjoining properties are operated together. The other two properties utilize narrow-gauge electric locomotive haulage. Table 5 gives data concerning the open pits that use conveyor-belt systems.

TABLE 4.—*Locomotives*

Pit No.	Grade, Per Cent		Haul, Miles	Lift, Ft.
	Maximum	Average		
STEAM				
1	3 comp.		2½	210
2		1.5	1	40
3	2.5	1.5	3½	150
4	2.5	1.5	3¼	165
5	4		2	422
6	3		3000 ft.	90
7	3		2¾	200
8		1.85	4	250
9	3	2	1½	195
10	3		3½	135
11	3	2	1¼	120
12	0		3¼	270
13	2.5	1.8	2½	225
14	5	2.25	4¾	457
15		3	1	110
16		1	¾	20
17		1.25	¾	26
ELECTRIC				
1	3	2	2½[a]	160
2	3	2	2½[a]	160
3	3		4/5[b]	32
4	Unreported			

[a] Standard gauge.
[b] Narrow gauge.

TABLE 5.—*Conveyor Belts*

Pit No.	Number of Flights of Belt	Belt Inclination	Width of Belt, In.	Length of Belt, Ft.	Speed of Belt, Ft. per Min.	Total Lift, Ft.
1	3	18°	36	900	400	265
2	3	19° 30′	30	865	365	144
3	9	11° 20′	30	4,480	500	386
4	3	15	30	961	500	176
5	3	15	30	966	500	201
6	6	15	30	938	500	228
7	Construction not completed					

Trucks are used as gathering units in six pits in connection with the belts and caterpillar-wagon units in one pit. These gathering units operate usually on easy grades which vary in inclination and increase as the pits near exhaustion. Table 6 shows the maximum and average grades together with the length of haul where trucks are used to haul the ore out of the pits. In these pits

TABLE 6.—*Grades and Length of Haul When Trucks Are Used*

Pit No.	Grade, Per Cent		Length of Haul, Ft.	Total Lift, Ft.
	Maximum	Average		
1	10	5	2,600	130
2	8	6	3,400	170
3	8		2,700	135
4	8		3,800	180
5	8	7	4,000	296
6	8	5	3,500	170
7	8		4,400	237
8	8		2,000	88
9	5	3	11,000	199

the ore is transported from the bottom of the pit, or loading point, by truck to railroad cars or plants on surface. That the heavy-duty truck is proving a serious competitor of locomotive haulage is shown by the following figures on the number and size of trucks utilized in ore haulage on the 32 properties reported:

Number of trucks....	1	4	93	2	7	2
Gross tons pay load..	22	20	15	12	9	4.5

While many of the unreported properties are using truck haulage, the capacity of most of these trucks is less than 9 tons, so they would be classed as light-duty rather than heavy-duty trucks. The 20-ton and 22-ton trucks have been limited to ore haulage, whereas heavy-duty trucks in sizes from 15 down to 9 tons as well as the lighter, standard units are used for both stripping and ore. It is evident that the 15-ton heavy-duty truck is by far the favored size, and it should be noted that all of the trucks listed above have hydraulically operated end-dump bodies.

In trucks carrying from 9 gross tons and up, the Diesel engine predominates and the supercharged Diesel leads the standard in number. The semi-Diesel, or oil-engine

trucks follow the Diesel in popularity in the larger units. Gasoline as motive power is limited to the lighter units.

Locomotives and Cars.—Tonnage of ore transported by locomotives and cars still maintains a respectable lead over trucks, largely because the extremely large pits are all equipped for locomotive haulage. Whether or not the heavy-duty truck will replace the locomotive and car equipment in the larger pits remains a question that only time and economics can determine. The distribution in number and size of steam locomotives utilized in ore haulage is shown in Table 7.

TABLE 7.—*Steam Locomotives*

Tons on drivers...	60	62	65	78	82	92	100	109	127	130
Number of units..	8	2	4	[illegible]	23	6	3	8	2	4

In the only open pit reporting standard electrified locomotive haulage, six electric locomotives are used, weighing 60 and 65 tons. Electricity is used in hauling to a shaft pocket in two open pits on the Range, one of these operations utilizing three 20-ton electric locomotives and the other three 10-ton. In the only open pit reporting caterpillar and wagon haulage, Diesel tractors are used in conjunction with 13-cu.-yd. side-dump wagons.

Tracks.—The approach tracks utilized in locomotive and car haulage of ore are not temporary, as in stripping, consequently these tracks usually are laid with heavier steel in a standard manner, well ballasted on the ore approaches to the pit. Owing to the frequent shifting necessary as shovels cuts are completed, the loading tracks usually are ballasted with ore material only whereas on the approaches gravel ballast is usually used.

Haulage Roads.—In the six pits utilizing trucks as gathering units feeding belt conveyors, the roads are entirely in the ore material and no surfacing is used. Of the 10 pits listed as having all truck haulage from the pit, 7 discharge their loads into railroad cars or crushing plants on surface and 3 into dump cars for haulage to washing plants. All except two of these pits surface their entire road system with ore material. Of the two exceptions, one pit utilizes a black top and one pit a stabilized road.

It is common practice in the open pits using truck haulage in the pit bottom, and also on the pit approach roads utilizing gravel or ore material as a surface, to make use of a sprinkler to avoid dustiness during summer operations. The roads are maintained either by means of a road patrol or by a tractor and blade.

General.—Many of the open pits using trucks maintain portable garages for servicing, usually on the crest of the pit near the approach, together with the necessary fuel tanks, air compressors for tire inflation, and general servicing tools. Where a changeover has been made from locomotive-car haulage to truck haulage, the former locomotive shops have become garages where major repairs and maintenance work are done.

Most of the operations were equipped with steam locomotive cranes for maintenance work at the shovel but since the tracks have been eliminated these cranes have been replaced by lighter units operating on caterpillars or on a truck chassis.

Great change in personnel has taken place with the changing types of haulage equipment, and field truck supervisors are to be found on most large truck jobs. Where conveyor belts are employed usually a belt inspector is in charge of the maintenance work for the purpose of checking up cuts and other defects to be repaired by means of the portable vulcanizing units.

It has been found necessary to train shovel runners to handle their equipment in such a way as not to damage the lighter type of haulage unit now in use. The old practice of bouncing rocks off the dipper teeth would now be punishable by layoff. It is surprising to see how well the shovel runners have adapted themselves and the

use of their shovels to the newer type of haulage equipment.

Underground Mining

Since shipments of ore can be made only during the Lake Superior transportation season, open-pit ore production is limited to five or six months of each year, but underground work can be carried on the year round. For at least half of the year underground production is stockpiled near the shaft collar and usually is loaded out early in the shipping season; for the other half year, underground ore is loaded directly into railroad cars for transportation to the dock. Most of the underground stockpile trestles are replaced each year because they are of timber and are destroyed during the loading out operations.

Top Slicing.—In top slicing the ore is removed by retreating from the ore limits toward the shaft. After one level has been mined back sufficiently to allow for safety, a sublevel is mined below the first, and so on, the result being stepped-back mining levels at the various elevations. All slices are timbered at 5-ft. or 6-ft. intervals, various heights of posts and length of caps being used on the sets.

From the sublevel drifts, usually slices are run out about 100 ft. at right angles on the long side and approximately 10 ft. on the short side. In some instances slices are run out to 50 ft. on each side of the sublevel drift.

The cycle of slicing is the normal one, which has often been described. Slicing is carried on by the contract system. Usually two miners per shift work on each slice, the payment being for the tonnage produced, based on car count. A minimum wage is set, with no maximum wage for miners. The miners usually perform the entire cycle of slicing.

Surface Buildings.—The surface buildings of an underground mine consist of a dry or change house with all modern conveniences for the cleanliness and health of the miners, engine houses containing electrically operated hoists, compressors and rotary convertors or generator sets for the direct current required underground, warehouses and other buildings. The powder houses are built at some distance from the others.

Timber Yards.—The timber used underground is stored in yards next to the timber shaft, or near the hoisting shaft when no timber shaft is used. No timber framing is done on surface on the Cuyuna Range but on the Mesabi and Vermilion Ranges all mines do at least a part of the framing on surface and one mine completes it.

Headframes.—Steel headframes are used on all properties reported except for one timber headframe at an incline shaft on the Cuyuna. Usually hoisting is done in counterbalance on ore and a single timber skip is used in the timber shaft.

Hoisting and Timber Shaft.—Hoisting and timber shafts are sunk through comparatively unstable surface material, with shaft sets of timber or concrete supported by hanging beams, usually at the collar and

Table 8.—*Shaft Data*

Hoisting Shafts	Cuyuna Range		Mesabi Range			Vermilion Range	
Shaft depth, ft.	480	450 I.[a]	250	280	312	650	1466
Depth to pocket, ft.	450	264 V.[a]	205	236	228	406 to 602	1360 to 15th level
Number of compartments	5	2	4	3	5	5	4
Size of shaft	11 × 13	6 × 14	13 × 16	6 × 20	11 × 18	11 × 18	10 × 24
Skip capacity, gross tons	3.65	2.5	4	3.7	5	5	7
Kind of shaft	V.	I.	V.	V.	V.	V.	V.
Number of timber shafts	2	1	1	1	3	None	None

[a] V., vertical; I., incline.

at the top of ore or top of ledge. Plank on concrete lath is used as a lining.

Only seven underground mines were reported, two on the Cuyuna Range, three on the Mesabi Range, and two on the Vermilion Range. Shaft data appear in Table 8. No underground mines have been opened on the iron ranges for several years, the last large shaft-sinking job on record being the shot-drilled, borehole shaft on the Vermilion Range.

Development

Drifts.—After the station for the main level is cut, the main ore-haulage drift is run out to the ore limits in one or more directions. If this main drift is driven in rock underlying the ore body, no timbering is required, but if it follows the top of the rock, or is cut in the ore, timbering is required. On the main-level drifts, raises are driven at desired points up to the highest operating level. Sublevel drifts are driven from the raises to the ore limits, and mining rooms or so-called slices are driven at right angles to these sublevel drifts as stated above.

Separate timber drifts are sometimes driven from timber shafts paralleling the main drift, following the lines of raises, in order to furnish timber to the working places without interfering with ore haulage. Of the seven underground mines reported, only three have separate timber drifts; in the others, the main level is used for haulage of both ore and timber.

Raises.—Raises from the main level up to the working places are driven in advance of mining, so that several raises are accessible to a working place. These raises are used for ore storage, as ladderways and as passageways for compressed-air lines, electric cables and ventilation tubes. The ore is drawn from the storage raises for transportation to the shaft, the ore flow usually being controlled by quarter-pan gates. Bars and compressed-air pipes are manipulated to prevent bridging of ore. Raises are also utilized for hoisting timber from the sublevel or main drift or timber drift to the working place.

Mining of Ore Underground

The sizes of the slices used at the various mines reported are listed in Table 9.

TABLE 9.—*Sizes of Slices*

Raise	Cuyuna Range		Mesabi Range			Vermilion Range	
	A	B	C	D	E	F	G
Height of post, ft.	12	12	15	10	8–12	9	10
Length of cap, ft.	14	10	12	10	11	9	11

Drilling for slicing is done largely by means of jackhammer augers in soft ores and by drifters in hard ore or rock development. V-point auger steel is used in jackhammers and detachable bits in drifter steel. Table 10 sets out the pertinent data on drilling required.

TABLE 10.—*Drilling Data*

Item	Cuyuna Range		Mesabi Range			Vermilion Range	
	A	B	C	D	E	F	G
Type jackhammer auger	RB12	RB12	RB12	RB12	RA & RB12	RA & RB12	Ra12
Number jackhammer augers used	18	15	38	17	41	93	67
Drilling rate, ft. per hr	100	25	300		45–60	45–60	30–20
Depth of hole, ft	6	6	6	6	6	6	4½–5
Number holes per round	12–16	13–21	12		12	12–15	14
Size air compressor, cu. ft. per min	1200	1400	2000	1375	1500	3221	2764
Air pressure, lb.: maximum	100	95	110	110	100	100	100
Minimum	70	80	70	95			

Blasting is done largely by the use of 60 per cent Gelex except in one Vermilion mine where 45 per cent ammonia Gelex is reported. Electrical detonators are used. Tons ore broken per pound of explosive are reported as follows: Cuyuna mine B, 1.2; Mesabi C, ⅔; D, 2; E, 1.73; Vermilion, F, 1.73; G, 1.94.

Blasting is done between shifts when possible but ventilation systems have been introduced so that this is no longer essential.

Where long posts and caps are used tuggers facilitate the placing of timber. Posts and caps are set in place, roof lagging placed between the caps to protect the back, and the entire set is tightened by means of wooden wedges. Sprags are placed between the sets, and the timbering is completed.

In all the mines reported, the ore, after blasting, was removed from the slice by electric tuggers and scrapers. These tuggers in general are operated by 25-hp. motors; a few 6½ to 10-hp. motors are used.

In only one mine is the ore removed from the slice and loaded by these scrapers into hand-tram sublevel cars for transportation to the raises. At another the scrapers deliver to large transfer scrapers, which transport the ore a distance of 900 ft. from the slice entry to the raise. Slice scrapers are used in another part of this mine to deliver the ore to shaking conveyors, which in turn deliver their load to conveyor belts 410 ft. long. At all other places the ore is delivered direct from the slice to the raise by the smaller scrapers. After removal of ore from the slice some hand clean-up work is necessary prior to the timbering operation.

In every mine reported the ore is transported from the raise to the shaft by means of electric motors in bottom-dump steel tram cars operating on narrow-gauge track.

Drifts are driven to provide drainage back to the shaft sump, where it is pumped to the surface. Flow of water varies as follows: Cuyuna, 950 and 900 gal. per min.; Mesabi, 15 gal., 75 gal. and 750 gal. per min.; Vermilion, 600 and 150 gal. per minute.

Main shafts and timber shafts connected by means of drifts and raises provide natural ventilation circuits. Secondary fans and tubing are provided to ventilate the working places on sublevel. Where ventilation through shafts and timber shafts is not sufficient, heavy-duty ventilating fans on surface are provided.

Miners work on a contract basis but all maintenance men—car loaders, hoisting engineers, surface men (timber handlers, and stockpile men), electricians, mechanics, motormen and timber repairmen underground—work on company account or wage system.

Underground Mining on Decline

Underground mining has been on the decline for several years, even before the latest depression. That no new underground mines have been opened even during the present emergency provides a definite assurance that underground mining costs in general must be reduced before new underground mines will be opened.

The open pits of Minnesota still have tremendous productive capacity and the needs of the present emergency can be supplied from that source. The reopening of underground mines must be postponed until there is a shortage of open-pit ore.

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DISCUSSION

(E. D. Gardner presiding)

S. A. Trengove,* Rolla, Mo.—The data Mr. Holt has obtained through careful investi-

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gation will be very useful to mining engineers everywhere, and especially to those with a feeling of personal interest in the practice in Minnesota mines.

Several techniques being used in some of these mines have been omitted in the paper. I believe these techniques should be commented on briefly in order to call attention to certain improvements in Minnesota mine practice that are rather important.

For instance, no mention is made of the use of the bulldozer in preparing the subgrade and final grade for the standard-gauge railroad tracks. It can be stated definitely that this method of preparing track grades has not only resulted in a saving of time, but has actually made possible smoother finished track grades with reduced shock to rolling stock. The bulldozer has also made it possible to lay tracks over rough territory in areas where tracks might not otherwise be given consideration, and this frequently makes it possible to attack the ore body at points that otherwise might have had to remain untouched for some time.

The wheeled scraper being used in considerable numbers in some mines for "clean-up" purposes should also be mentioned as an improvement in the final phases of mining work. In many of the open-pit mines it is difficult to do a thorough "clean-up" job with small shovels loading into railroad cars. One practice in the past has been to utilize a small shovel loading directly into railroad cars on track laid on the rough bottom. Such practice necessitated what might well be considered an excessive amount of track laying per ton of ore, in addition to what was often an over large amount of locomotive spotting time. The shovel frequently was idle between train trips.

Using a type of railroad loading ramp similar to that described for certain truck operations, the wheeled scraper has been successfully applied to the clean-up problem. The iron ore is especially suited to handling by this method, since it is heavy enough, and of the proper consistency to flow into the scraper bucket and fill it well. Both rubber-tired and crawler-type scrapers are being used, with box capacity ranging from 3½ to 7 cu. yd. Several different designs of scrapers are employed, varying somewhat in general design and in the required hydraulic pressures used.

A most important advance in blasting practice has also been made in some mines in the Mesabi district. This concerns the use of ammonium nitrate explosive, using Primacord as the detonating agent in big blast-hole practice. The safety features of this method for open-pit blasting have a special appeal. The explosive is very safe to handle, as is also the detonating agent. The use of the method, although requiring a little more care than other types in loading (owing to the necessity of getting the cans of explosive into good contact) offers a high and unusual degree of safety; that is to say, the region of the blast is completely safe until all loading work has been done. All men and materials can be removed from the scene in complete safety before the electric blasting cap is finally applied to the main trunk line of Primacord. After this the blast may be set off in complete safety.

The large auger drills mentioned in the paper are being extensively employed for drilling horizontal holes into the base of ore and stripping banks, and have added something in the way of drilling speed for the softer materials. The type of drilling now in use employs a mole's foot bit with replaceable cutting claws. This tool not only permits a more rapid drilling of horizontal holes of large diameter but also offers a new line of distribution for the loading charge. The charge is now deployed horizontally instead of vertically, as with churn-drill holes. For some banks, the method is believed to be of increased efficiency over the vertical method of placement of the charge.

In general, the employment of trucks and tractors and conveyor belts in the open-pit mines has done two important things:

1. It has furnished more rapid methods for mining much ore formerly delegated to "scramming" and "clean-up" operations.

2. It has reduced the amount of ore that would be tied up in track benches if the railroad method of approach were used.

Much of the haulage equipment now being used in the iron mines was developed in the earthwork excavation field for the purpose of handling material much lighter than iron ore. Some of this equipment has entered the mining operations without special reenforcement. This practice involves the necessity for considerable reenforcement of the structural members of trucks, tractors, etc. At the present time, however, equipment is being made available to

better withstand the heavy work of open-pit mining operations.

J. A. Carpenter,* Reno, Nev.—The past years have witnessed in the West the increasing use of opencut in preference to underground mining in the production of the base and precious metals. This is due to the lowered cost of moving rock through the rapid development of the traveling-tread Diesel-powered units motivating huge scrapers, blades and rooters.

Road-building contractors use these same units and a present-day tendency with many of the smaller opencut mines is to contract the stripping and mining of their ore bodies to these firms, thus saving the equipment investment and securing efficient operation.

To secure a steady feed to the reduction unit in opencut work during bad weather, it is often necessary to stockpile a reserve. This has led to the truck delivery of the ore to a flat area above a relatively small coarse-ore bin with the contractor's blade to feed the ore to the bin as needed.

One Nevada installation on the Comstock carried this idea further by contracting the reduction of the ore to ball-mill feed size through the contractor's traction crushing unit, using a cone depression in a waste dump as the fine-ore bin, thus having no capital investment ahead of the ball mill.

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* Director, Mackay School of Mines, University of Nevada.

Modified Mining Methods in the United Verde Mine

By J. B. Pullen,* Member A.I.M.E.

(New York Meeting, February 1941)

The United Verde mine is in the north central part of Arizona, on the northeasterly slope of the Black Hills, near the town of Jerome. Ore was first discovered in the district about 1875, and the first mining claims were located by M. A. Ruffner and associates in 1877. In 1888 the late Senator William A. Clark secured control of the property and under his ownership it developed into one of the country's most important copper producers. The property was sold to the Phelps Dodge Corporation in 1935, and since then has been under this Corporation's management. Except for shutdowns caused by general world-market conditions, production has been continuous from 1890 to date.

Geology

The mineralized zone ranges in area from approximately 12 acres to less than 5 acres. All mineralization occurs in the pre-Cambrian intrusives. The first intrusion, according to geologic records, was the quartz porphyry, which later was intruded by augite diorite. Following the diorite intrusion through a series of intrusions of lesser magnitude, a pipelike mass of massive sulphide, quartz and rock was formed. The diorite forms the hanging wall of this mass, and the porphyry the footwall. Between the footwall and the sulphide a band of chlorite schist was formed, probably owing to metamorphic conditions. Commercial ore occurs along the contact of the schist and sulphide extending into the sulphide, sometimes entirely through the schist, and fingering out into the porphyry. The ore deposit was formed by replacement similar to deposits formed in limestone areas.

Faulting and deformation occurred in pre-Cambrian times and again in the Tertiary period, removing approximately 2000 ft. from the top of the main United Verde ore body, forming the United Verde Extension ore body, the only other major ore body in the district.

Major development work has shown that the sulphide mass extends to the 3300-ft. level, and diamond drilling below this level has indicated that the mass might extend below the 3700-ft. level, but in a diminished quantity. While many copper minerals exist in the mine, the chief commercial mineral is chalcopyrite.

A plan and vertical section of the mineralized zone showing the principal geological eatures is shown in Figs. 1 and 2.

General Description and Physical Condition of Mine

After over 50 years of active production, ore still exists on almost every level of the mine. This condition is due to many causes —mine fires, which broke out as early as 1894, caused the abandonment of certain areas; subsidence and ground movements retarded the cleaning up of others. Furthermore, in the early days there was such an abundance of ore that when mining became difficult it was a simple matter to replace the production by going to a deeper level.

Several attempts were made to recover the ore in the fire zone, both by glory-holing

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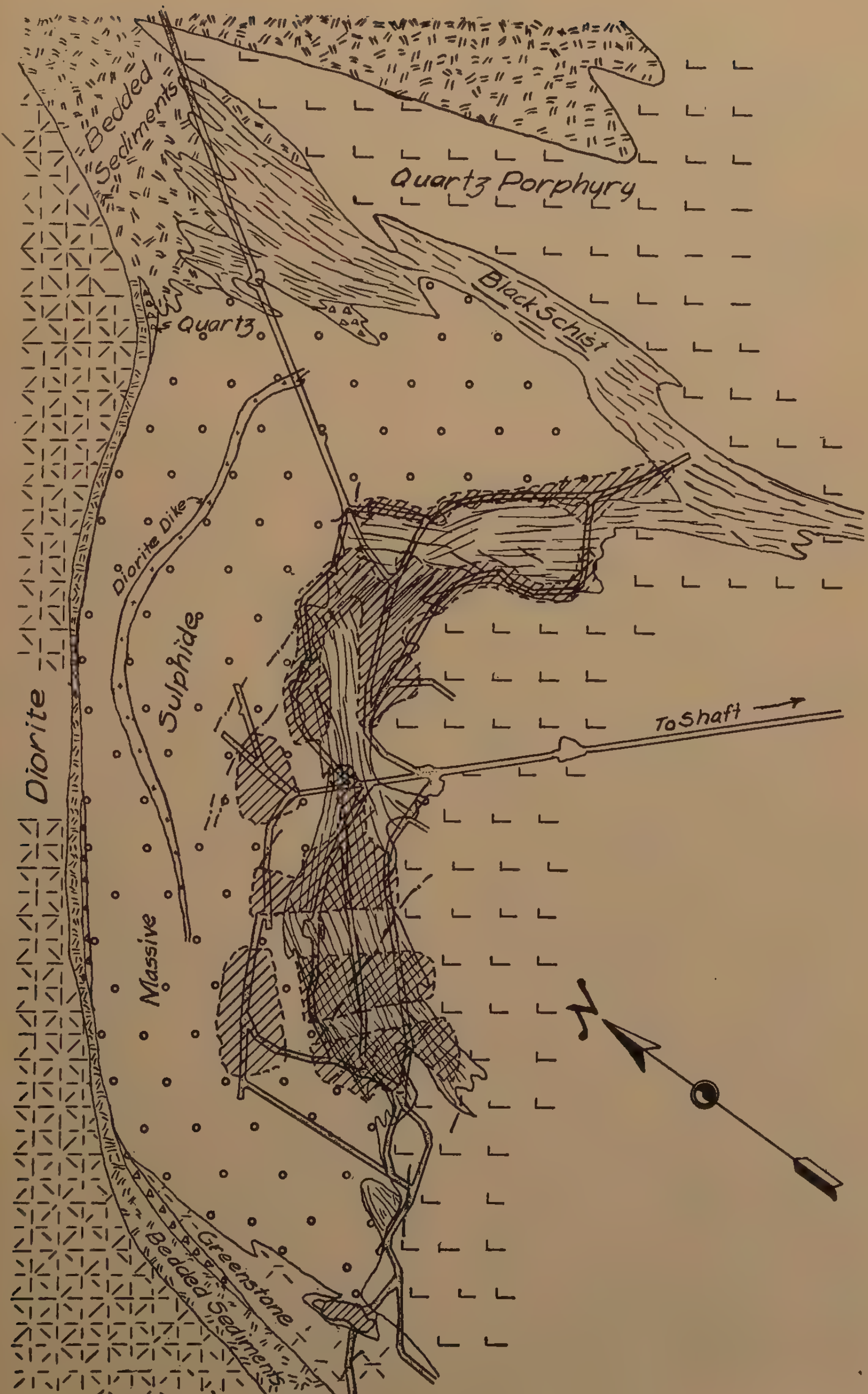


FIG. 1.—MINERALIZED AREAS-PLAN.

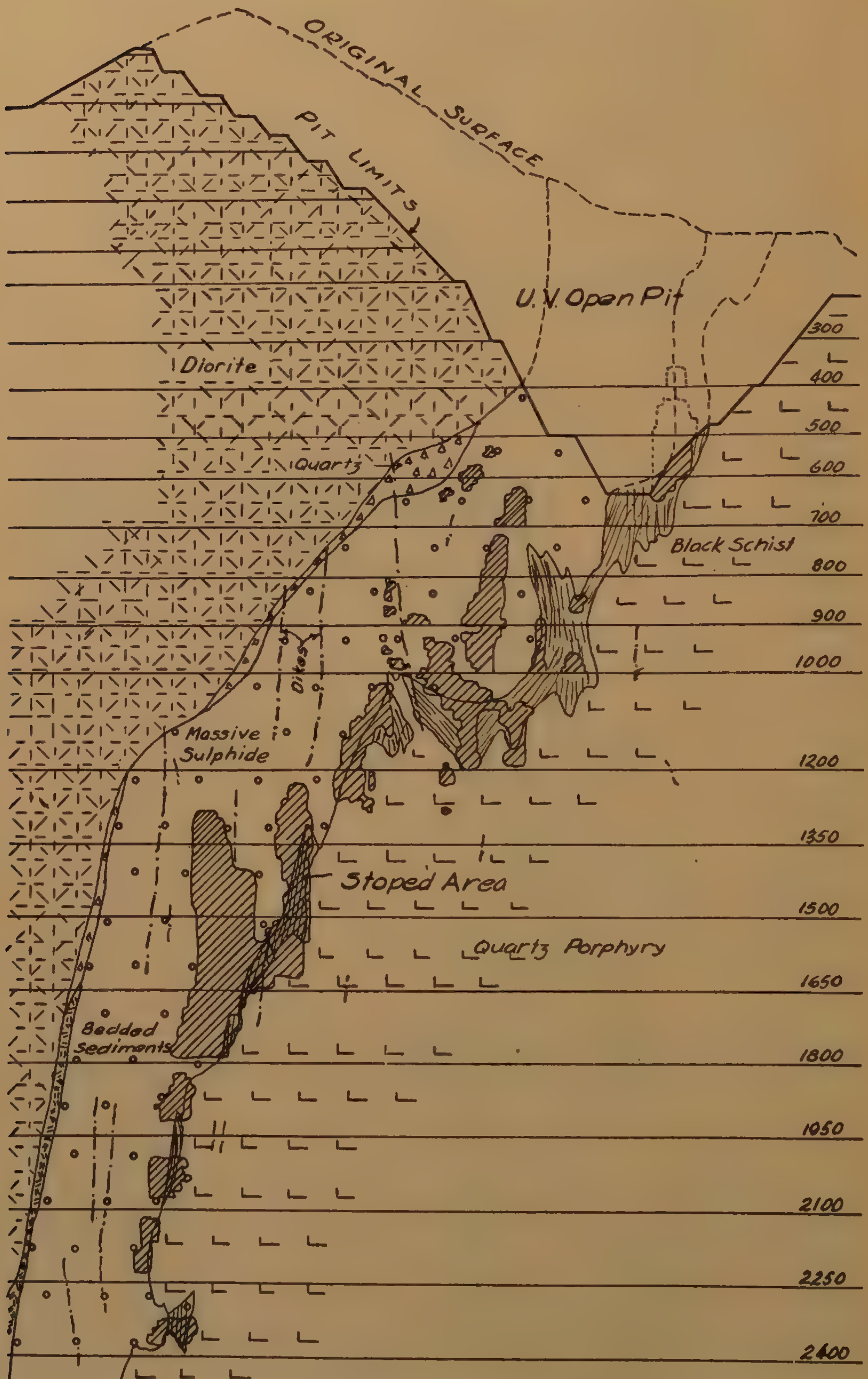


FIG. 2.—MINERALIZED AREAS-SECTION.

from the surface and by underground methods, without success; then plans were made to recover the ore in the upper portion of the mine by open-pit methods, using steam shovels and railroad equipment. Later this equipment was replaced by electric shovels and gasoline trucks. The open-pit mine was started in 1918 and was completed Apr. 1, 1940. The pit produced over 9,000,000 tons of ore requiring the removal of over 32,000,000 tons of other material. It was bottomed on the 650-ft. mine level and there still remain some remnants of ore in the walls, which will be mined by underground methods.

The underground mine extends from the 160 to the 3300-ft. level, with operations on every level from the 500 down (Fig. 3). Two main operating shafts are required, both of which are interior shafts and have underground hoist rooms. No. 6 shaft is used for the handling of men and supplies and extends from the 400 to the 3000-ft. level. It is connected with the surface plant by a 1600-ft. adit. The shaft is equipped with a counterweighted, double-decked cage that will handle 100 men or eight standard mine trucks per trip. Timber, steel, and supplies loaded on the surface can be handled on the cage and distributed to the various mine levels without rehandling. No. 5 shaft is used for the hoisting of ore and extends from the 800-ft. level to approximately 200 ft. below the 3300-ft. level. All ore below the 1000-ft. level is hoisted and dumped into ore bins on this level. Ore above this level is dropped down through ore passes to bins on this level. All ore is transported on the 1000-ft. level, which connects to the surface through the Hopewell tunnel. This tunnel is approximately 9000 ft. long and is equipped with a standard-gauge railroad track, over which ore trains, each made up of a 25-ton locomotive and eight 40-ton cars, are transported.

Waste fill for the mine is produced from the northeast wall of the open pit. This material is dropped through ore bins to the 1000-ft. level and transported to the main mine-waste system by the standard-gauge haulage equipment. The waste system is a series of raises extending from the 1000 to the 2550-ft. level with pull-off points and transfers on each level.

The level interval in the mine is 100 ft. from the surface to the 1000-ft. level, 200 ft. between the 1000 and 1200-ft. levels, and 150 ft. from the 1200-ft. level down.

No pillar support was planned for the mine until it had reached the 1500-ft. level. From this horizon to the bottom level, a series of both vertical and horizontal pillars has been carried. Extensive stoping operations have been prosecuted from the top level to the 2700-ft. Main-level development has been completed to the 3300-ft. and diamond drilling to the 3700.

Mining is divided into three major problems. Between the 160 and the 1500-ft. level the ore remaining is in the form of remnants left around incompleted stoping sections. From the 1500 to the 2250-ft. level the ore is in the form of vertical and horizontal pillars. From the 2250 level down the ore is in virgin ground.

In the mined areas extending from the 1200-ft. level to the pit bottom, subsidence and ground movement set in during 1929, which ultimately resulted in dropping all levels approximately 5 ft. Subsequent mining has required the redevelopment of all these levels.

Until recent years only feeble attempts were made to mine the ore left in the pillar sections, no real provision being made for its recovery. The underground portion of the mine was shut down from 1931 to 1937—at first because of market conditions and later awaiting the completion of the open-pit mine. It was thought that there was a possibility that the underground mining might affect pit operations. When work was resumed in the underground mine in 1937 over 70 per cent of the ore reserve was in the form of pillars and remnants.

Physical Characteristics of the Rock

As may be surmised from the geological description, ore occurs in three main types of rock: massive sulphide, schist, and porphyry. The sulphide ore is dense and hard, and is ideal for almost any type of an open-stope method. The schist is variable in texture; it is cut by slips and fractures and often is heavy and blocky. In previous years much of this type of ground had to be mined by some timbered method. In recent years, by holding the stopes down to a reasonable size and keeping the backs well arched, virtually all virgin ore is mined

Fig. 3.—Vertical section showing pillar arrangement.

by open-stope methods. The porphyry area may be classed as intermediate between the schist and sulphide. Except in shear zones, very little trouble has been experienced in mining by cut-and-fill methods.

Mining Methods

Prior to 1931, horizontal cut-and fill stoping, supplemented by square sets in heavy ground, was used almost entirely. These methods required the removal of ore by hand shoveling, the introduction of waste into the stopes by hand tramming, and hand spreading. When the mine was reopened in 1937 it was apparent that a considerable saving in operating costs could be made if it were possible to redesign the stoping methods so as to take advantage of gravity, and to use slusher hoists and scrapers.

In the lower levels of the mine the problem was comparatively simple, but for the recovery of the pillar sections and the remnants in the upper levels considerable experimentation was necessary. In the virgin areas on the lower levels, the problem resolved itself into the conversion of the large, flat cut-and-fill stopes into incline stopes, and as new stopes were silled out they were arranged for an incline method. For the pillars and remnants in the upper levels a modified Mitchell-slice system was adopted.

A Mitchell Slice

The Mitchell-slice system is not a new method. It was used extensively by the Calumet & Arizona in its Bisbee mines, and by the Cananea Consolidated Copper Co. at Cananea. As employed in the United Verde mine, the system consists of two parallel slots with square-set timbers 6-ft. square by 8-ft. high. These slots are driven at either 18-ft. or 24-ft. centers, depending on ground conditions. They are carried up to the top of the ore, the mat of the old stope above or to the desired height for a stoping section. After the slots have been driven to the desired height a connection is made between them and stringers are installed from slot to slot. Sufficient ground is then taken out so that stringers and knee braces can be installed at the joint next below of each square set. The ore remaining in the pillar between the slots is then stoped out by underhand methods and extracted through the slots, stringers being installed at every joint of the square-set section between the slots. Catching the top of the stope and installing the two top rows of stringers is the most important and most hazardous part of the operation. However, control methods may be used during this installation. Temporary support can be given the back by stulls, booms, or other props, while the top stringers are being installed. While the ground is being removed to install the stringers below, the top stringers can be supported by temporary stulls or blocks. In case of side pressure, angle braces may be used between the stringers at the pressure point. The length of these stopes has been limited to 60 ft. and the height to 54 ft.; a stoping section 48 ft. long by 42½ ft. high being the average. Wherever conditions will permit, 18-ft. stringers are used, but in extremely broken ground slots are installed at 18-ft. centers, which permits the use of 12-ft. stringers. All square sets are built of 10 by 10-in. timbers, using 12-ft. 3-post caps where possible. When ground conditions will not permit so large an opening, the cap is reduced to 6 feet.

The advantages of a Mitchell-slice system over a square-set system are: The Mitchell-slice system provides a means by which ground with a weak hanging wall can be mined with a minimum of unsupported back exposure. This system requires less timber than the square set and yet gives the same degree of support. It also provides ample room for the use of small slusher hoists and scrapers. After the preparatory

work, such as driving the slots and the installation of the top stringers, has been completed, the ore can be mined so efficiently that the high preparatory ex-

A typical Mitchell-slice section is shown in Fig. 4. Details of the timber framing and construction for square sets used in the slots are given in Fig. 5.

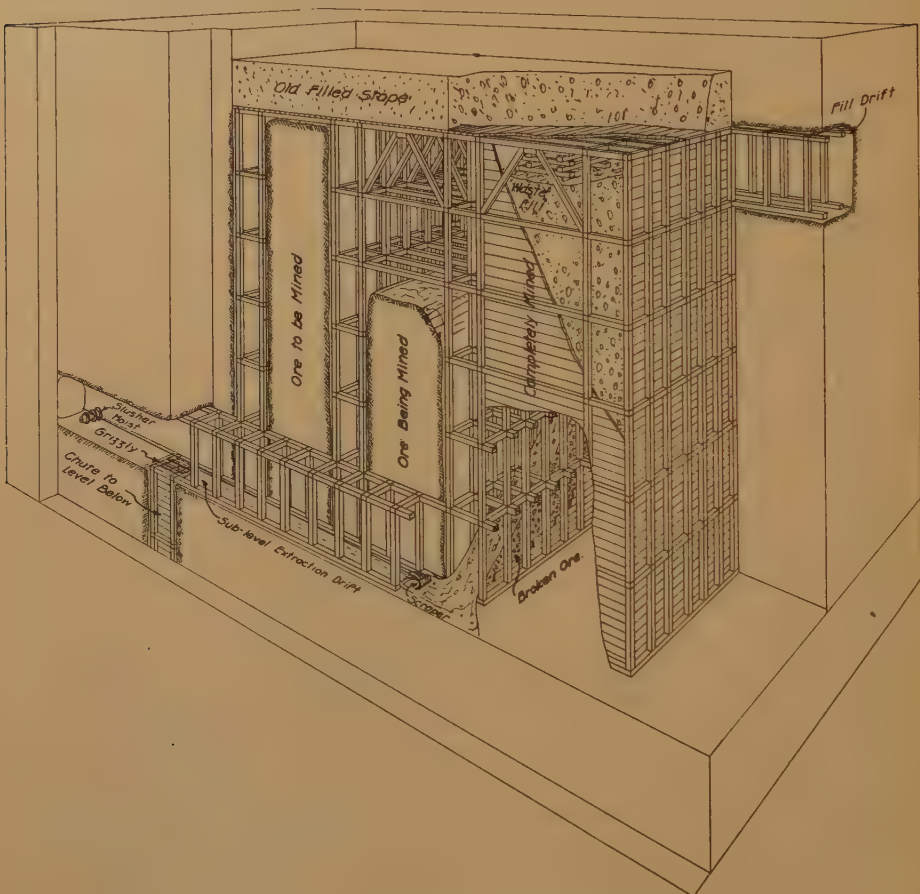


FIG. 4.—PERSPECTIVE OF MITCHELL SLICE.

pense is offset. In ground that tends to be heavy the area frequently can be completely mined and filled before the timbers have had time to give way. However, the system does require the opening of a considerable space at one time and a quick and adequate source of fill must be available. As the fill advances to the height of the stringers, the latter may be removed. They have some salvage value and by their removal a denser and tighter fill is obtained.

APPLICATION OF MITCHELL SLICE

Vertical Pillars.—The vertical pillars as maintained are approximately 35 ft. wide and average 125 ft. in length between the footwall and hanging wall, extending from level to level. Experimental development work disclosed: (1) The mapping of the mined areas around the pillars was rather inaccurate; (2) the pillar lines had not been strictly adhered to; (3) pillar lines had not

always been protected by fencing; (4) in some places the fences had rotted out, leaving no protection; (5) the mats of the old stopes above the pillars had deterio-

Therefore, the pillars were divided into 50-ft. vertical lifts for mining between levels. It was also found that better support could be given against the fills in the old

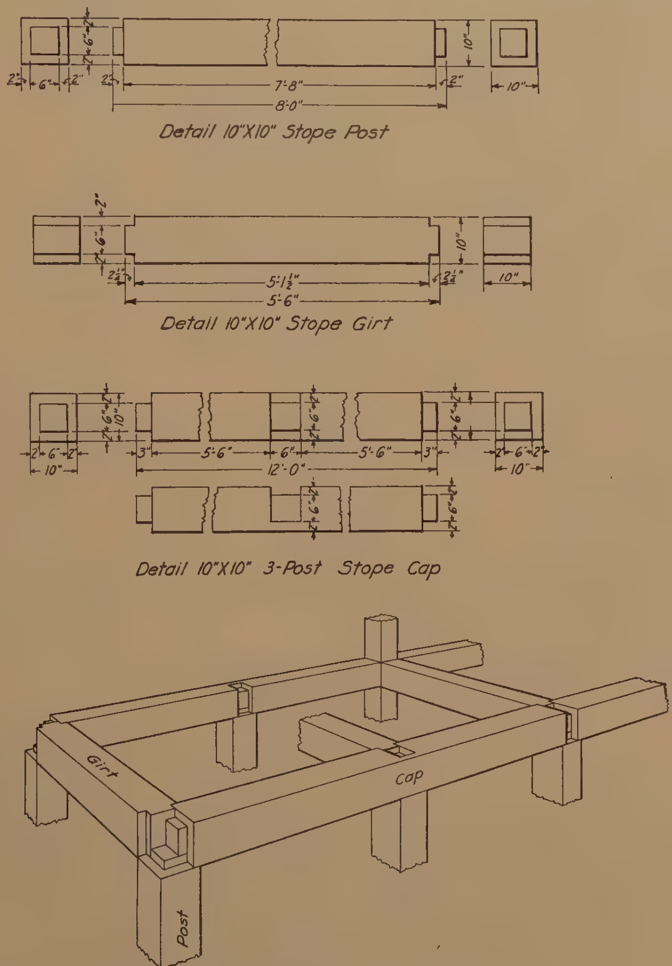
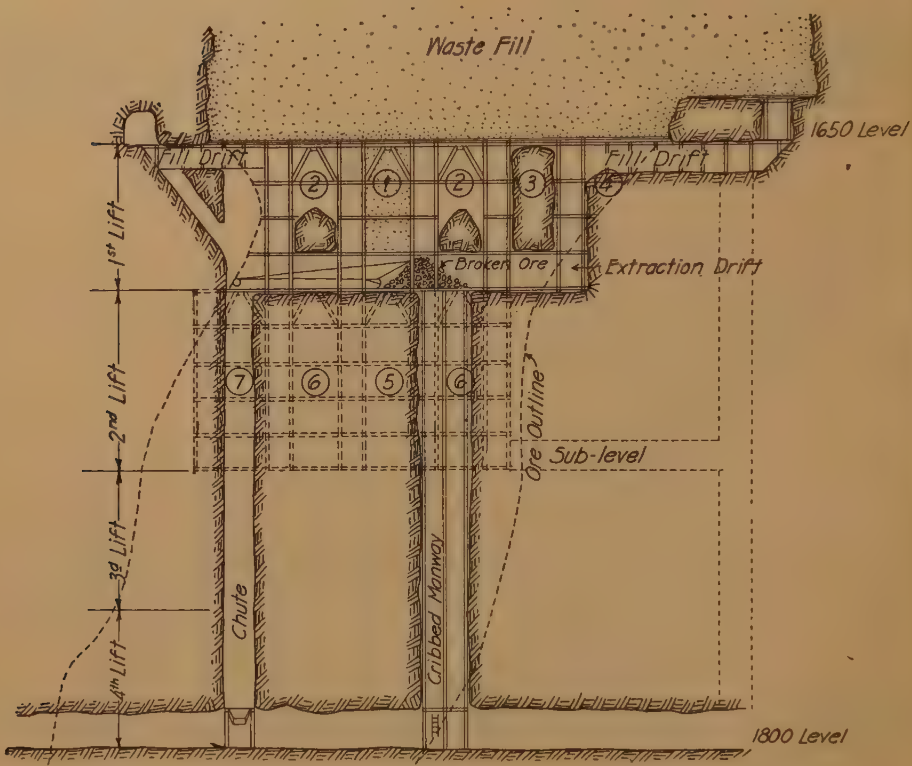


FIG. 5.—LEAD SET TIMBERS.

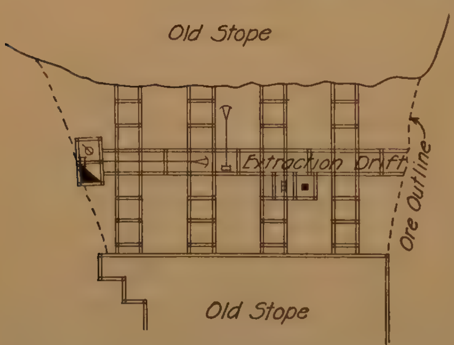
rated; (6) it would be necessary to provide extraction chutes, manways, supply entrances, and fill holes in the pillars themselves while actual extraction was in progress.

It was found that in applying the Mitchell Slice system to the vertical pillars a stope approximately 50 ft. high was about the maximum that could be handled safely.

stopes along the sides if the sections were laid out across the width of the pillar. This has proved also to be the best method for supporting the mat below the filled areas above. In most cases approximately 450 ft. of fill must be supported above the old stope. With the Mitchell slice running across the width of the pillar, one side of the slots is usually against solid ground and



Longitudinal Vertical Section
(Circled Numbers Indicate Mining Sequence)



Plan at 16th Floor Sub-level

Fig. 6.—Mitchell slice applied to vertical pillar.

there is very little buckling effect on the stringers between the slots.

In laying out the Mitchell-slice sections the first operation is to provide extraction chutes in the pillars. Where raises had been previously driven through the pillars, these are used, as there is not sufficient ground left in the pillar to drive new raises that would be better located. After the extraction raises have been provided on the first lift, which is approximately 50 ft. below the top of the pillar, a square-set heading is run through the center of the pillar lengthwise. This heading provides access for the driving of slots across the pillar and for scraping ore extracted from the slot and out of the slices themselves. From this heading two parallel slots are started at either 18-ft. or 24-ft. centers in the middle of the pillar. The slots are carried up to the top and the slice completely mined and filled before additional sections are started.

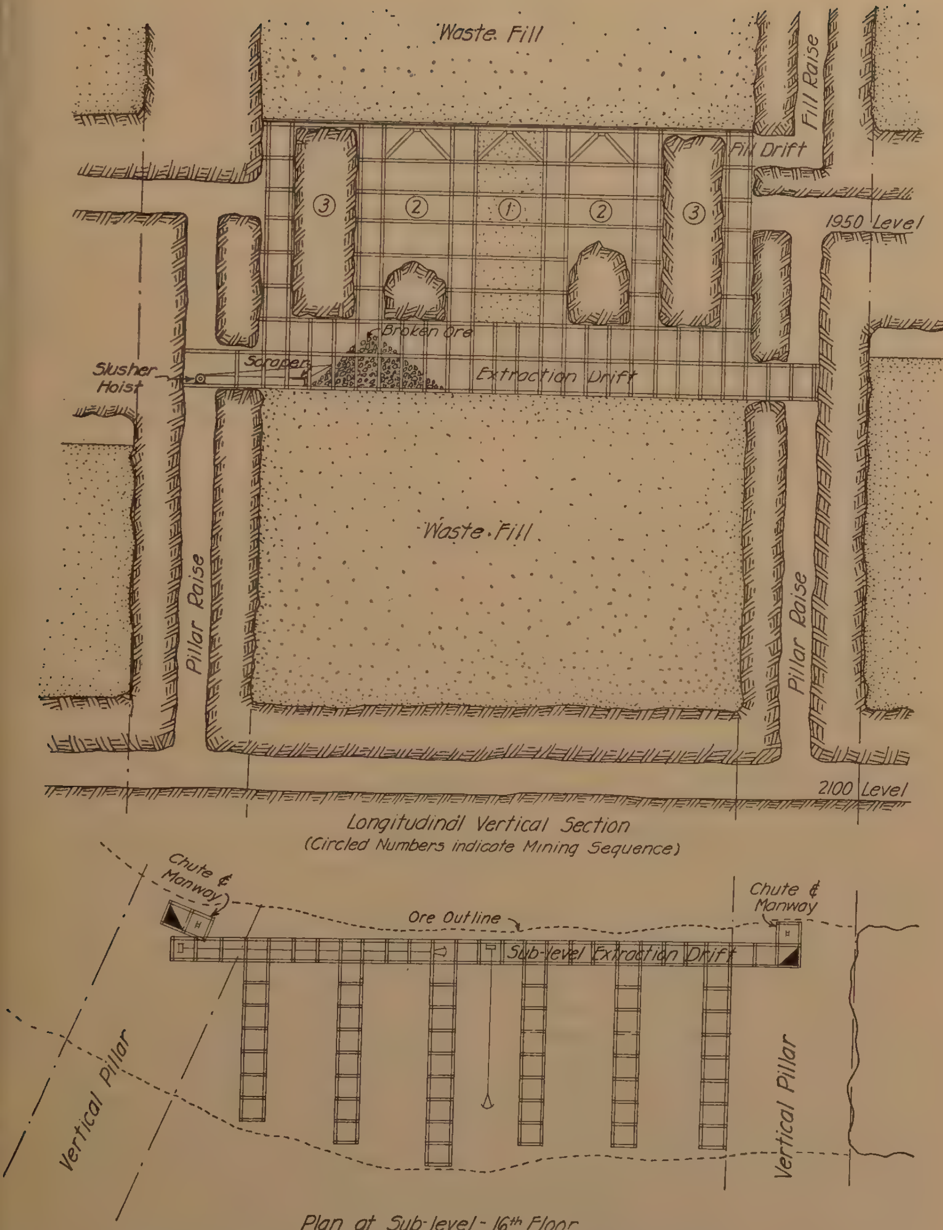


FIG. 7.—MITCHELL SLICE APPLIED TO HORIZONTAL PILLAR.

After the middle section has been completed, sections on each side are taken up and completely mined and filled. This process continues until the top lift has been entirely completed.

Fill for the section is provided by driving a heading under the mat through the length of the pillar. This heading is timbered and held open after the slices have been filled. Fill holes are kept open through the mined pillar by timbering around one of the square sets in the slot to provide fill for the lift below. The next lift is started and mined in the same manner and each lift is completed before mining is advanced to the lift below. The features of this application are shown in Fig. 6.

Horizontal Pillars.—The horizontal pillars are a block of ground 25 ft. below the level and 21 ft. above the level. The pillars average 80 ft. in width between the hanging wall and footwall and are about 150 ft. long, supported by vertical pillars at each end. The stopes below and above these pillars have all been mined and filled. The chutes in the old stopes were carried up through the fill and because of their age almost all the timbers had rotted out and the chutes had collapsed, therefore it was necessary to provide extraction chutes and fill raises out of each end of the vertical pillars. The feature of this application is shown in Fig. 7.

A square-set gangway was installed on the top of the fill of the old stopes connecting to each extraction raise out of the vertical pillars. These gangways were about 150 ft. long. Starting in the center of the horizontal pillar and working from the hanging wall toward the footwall, Mitchell-slice slots were started at either 18-ft. or 24-ft. centers. The center slice was carried up and completed to the mat of the old stope. Succeeding slice sections were then laid out for mining on each side of this vertical section until the entire pillar had been extracted between the vertical pillars.

Conversion of Horizontal Cut-and-fill Stopes to Incline Cut-and-fill Stopes

The horizontal cut-and-fill stopes that were converted to incline cut-and-fill stopes had been mined and filled for several floors. The chutes for these stopes had been carried up through the fill and all haulage drifts on the level had been arranged to accommodate them. In converting these stopes (Fig. 8), many of which were more than 100 ft. wide and up to 160 ft. long, each stope was divided into two sections, one section to be carried up and completed before the next section was started. The dividing line for the section was determined more or less by the chutes. Chutes were selected that could be placed in good condition by repairing. At least two chutes were used in each section. On top of these chutes a row of square sets was installed across the entire width of the stope to serve as extraction chutes, manways and supply entrances. The fill raise was then driven out of the end of the stope and along the ore contact to the level above. All other chutes in the section were bulkheaded at the top of the horizontal pillar, and filled. The back of the stope was then sloped up to the fill raise in successive cuts until about a 37° incline was reached, which was about the normal angle of repose for waste fill.

After the incline was established, successive cuts were taken, approximately 10 ft., starting at the square-set chutes and advancing toward the fill raise. The stopes are filled after each cut and floored with 2-in. planks. The square-set chutes are raised with each cut and gob fences are built along the pillar and section lines with a single line of square-set posts and caps. The installation of square-set chutes across the entire width of the stope made an ideal condition for slushing operations.

Silling Out New Stopes

New stopes are silled out at approximately 12 ft. above the rail. Usually drifts

have been driven, parallelling the contact, and the ore body inclines along the contact at about 60°. This makes it necessary to install the extraction chutes at right angles to the contact, so they can be stepped off and follow the pitch of the ore. The stope usually extends on both sides of the haulage drift.

The first operation is to bell out on top of the drift so as to leave a pillar of ground about 10 ft. thick on the level. The belling out is done usually with stoper machines and the ore is allowed to be dropped down onto the track and is loaded into cars by a mechanical loader. After the belling-out process has been completed, gangway timbers are installed on the level at 6-ft. centers, and stringers, 10 by 12 in. or heavier, long enough to rest on the ground pillar on each side of the drift, also are installed. The stringers are floored with two thicknesses of 3-in. planks. The stope is then completely silled out to the ore

and a fill raise provided, square-set chute timbers are installed on the side opposite the fill raise at right angles to the drift. The broken ore is drawn out of the stope, the pillar protected with sill timbers upon which a mat of old scrap wood is placed, and the stope filled. Successive 10-ft. cuts are taken up and the stope is filled and floored after each cut. This method is explained in Fig. 9.

General Notes on Stoping Operations

With the use of Mitchell-slice sections and of the smaller cut-and-fill sections, operating crews are broken down into small units ranging from two to five men in each stope. To use such small crews, it was necessary to develop the men into the all-around miner type. At present there is very little division of labor; i.e., the drilling, timbering, filling, and slushing operations are all prosecuted by the same crew.

Plan of Typical Stope Showing Locations of Chutes and Fill Raise

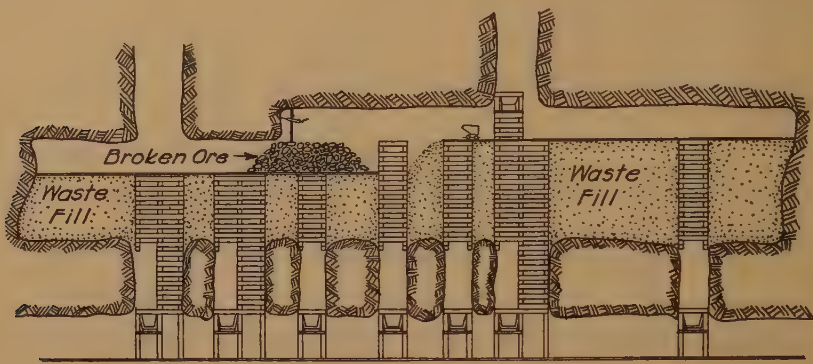
Fig. 8.—Conversion method.

boundaries and inclined toward the fill-raise end. A fill raise is driven out of the back of the stope or, where possible, a Calyx hole is drilled down from the level above. After the initial cut has been made

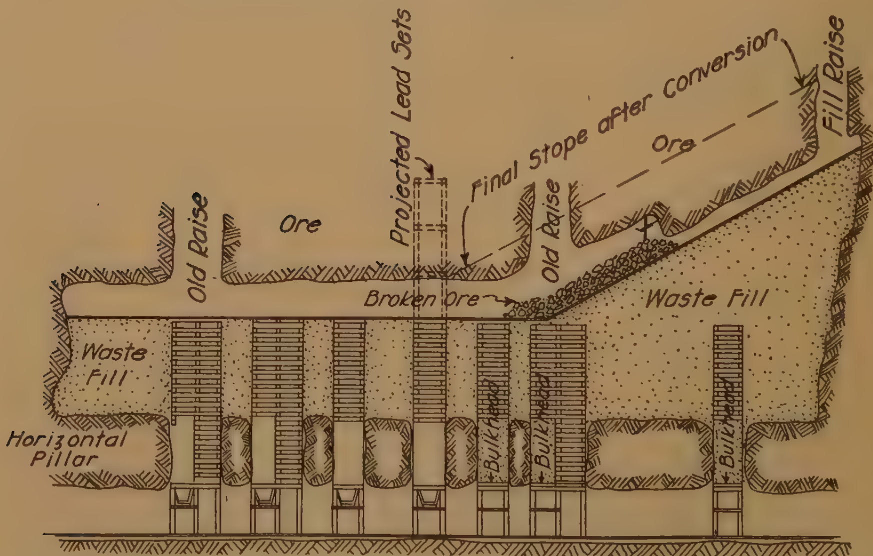
Detailed stoping statistics are given in Table 1. These costs include labor, timber, and powder for the breaking of the ore to the delivery in the chutes, plus the spreading and handling of waste after it has been

hauled and dumped into the fill raise or stope.

the usual raising and winzing methods for ore chutes, waste chutes, ventilation raises,



Typical Section of Horizontal Cut and Fill Stope Before Conversion to Inclined



Typical Section Showing Conversion Proceedure from Horizontal to Inclined Cut and Fill

Fig. 8.—(Continued).

Useful Operating Ideas

Calyx Drilling.—A Calyx drill (Fig. 10) has been used in the underground operations to some extent as a substitute for and as a pilot hole for winzing and shaft sinking where a level connection was available or existed at the bottom of any proposed shaft or winze. The drill used drills a 48-in. hole to a depth of about 400 ft.

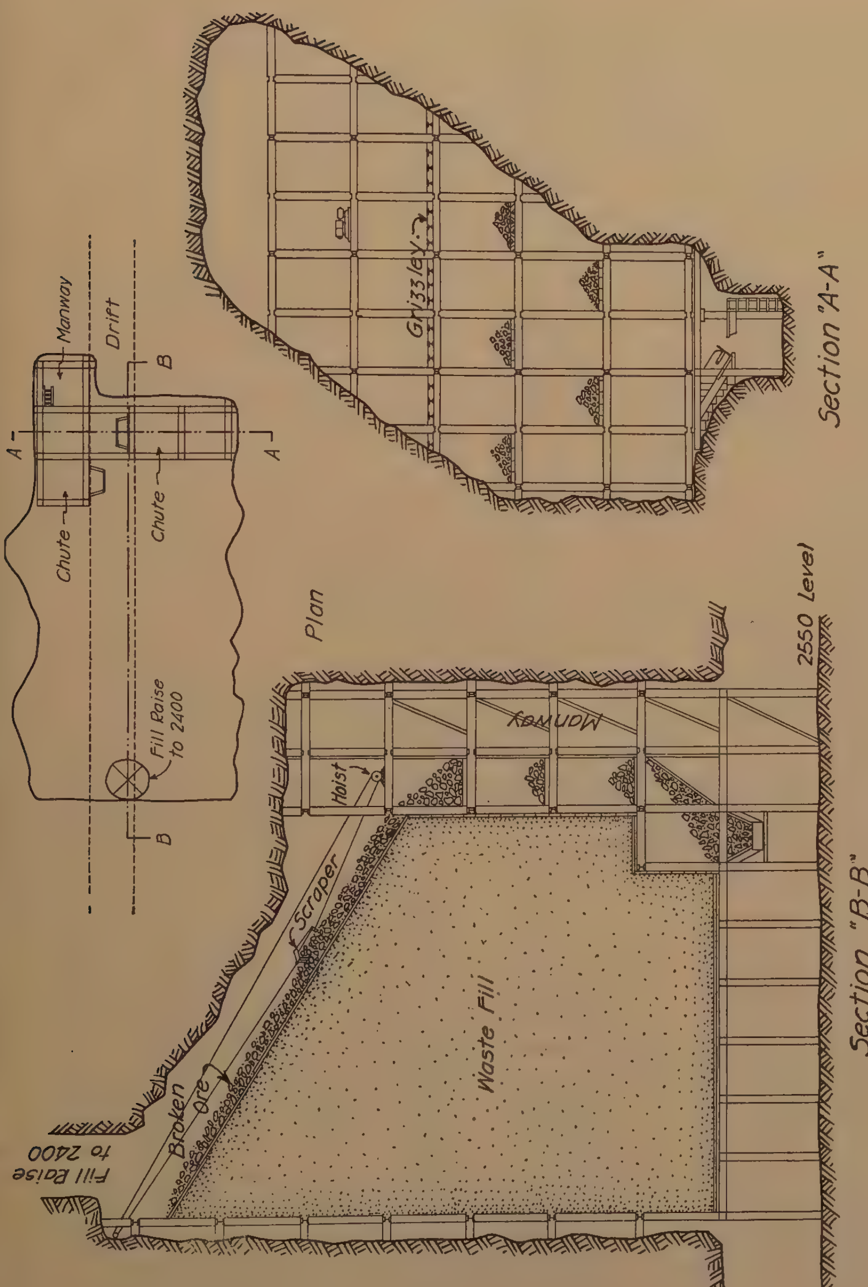


FIG. 9.—SILLING OUT NEW INCLINE CUT-AND-FILL STOPE.

It was placed in operation in October 1937 and has drilled 10 holes with a total footage of 1450 ft. The cost of holes more than 100 ft. deep has been lower than average drilling short holes (100 to 200 ft. long) for completely delineating the ore in any area before stoping operations are started. These light rigs have also been used in

TABLE 1.—*Stoping Costs and Efficiencies*

JAN. 1, 1940 TO AUG. 31, 1940

Tons Mined	Cost per Ton				Pounds Explosives per Ton	B.F. Timber per Ton	Tons per Man-shift
	Labor	Timber	Explosives	Total			
700 16-G Stope—Typical remnant of uncompleted stope							
7,233	$0.537	$0.235	$0.078	$0.850	0.44	7.47	10.46
1800 No. 2 and No. 3 Stope—A Typical Vertical Pillar							
7,100	0.578	0.184	0.061	0.823	0.35	5.89	9.06
2100 9-R Stope—Typical Horizontal Pillar							
9,260	0.624	0.287	0.160	1.071	0.93	9.26	8.74
All Mitchell-Slice Stopes							
97,157	0.589	0.224	0.090	0.903	0.52	7.18	9.35
Inclined Cut-and-Fill[a]							
177,019	0.462	0.128	0.136	0.726	0.78	4.08	11.87

[a] All stopes on the 2400, 2550 and 2700-ft. levels.

raise costs. A three-man crew is needed per shift. No. 8 steel shot is used as the cutting medium. Drilling data for average ground are given in Table 2.

TABLE 2.—*Calyx Drilling Data*

Operating Data	Average Ground
Depth of holes, ft. (maximum)	281.5
Footage per drill shift	2.76
Drilling speed, in. per hr.	14.1
Drilling time, percentage of total	20.4
Pounds shot per foot hole	22.8
Bit wear, in. per 100 ft. hole	6.6
OPERATING COST PER FOOT	
Labor	$ 6.62
Calyx shot	2.92
Supplies	1.13
Timber and explosives	0.18
Electric power	0.25
Total cost per foot	$10.10

Short-hole Diamond-drill Work.—Light, portable diamond-drill rigs are used for proving up areas carried as ore reserves around old stoping sections. They have provided a much quicker and cheaper method than the ordinary drift and raise methods.

Scrapers.—An all-welded scraper (Fig. 11) has been designed and built in the company shops. It may not be suited to conditions in other mines, but has been highly successful in the conditions at the United Verde mine. Scrapers in these stopes must have excellent digging qualities, to handle coarse ore and at the same time be suited for the spreading of waste.

The scraper is designed and weighted so that it will not turn over during normal operations, and it is always in an upright position. The teeth can be removed easily when the scraper is to be used as a waste spreader. Only one bolt is required for the fastening of the teeth. Both ends of the teeth are hard faced and pointed, so that

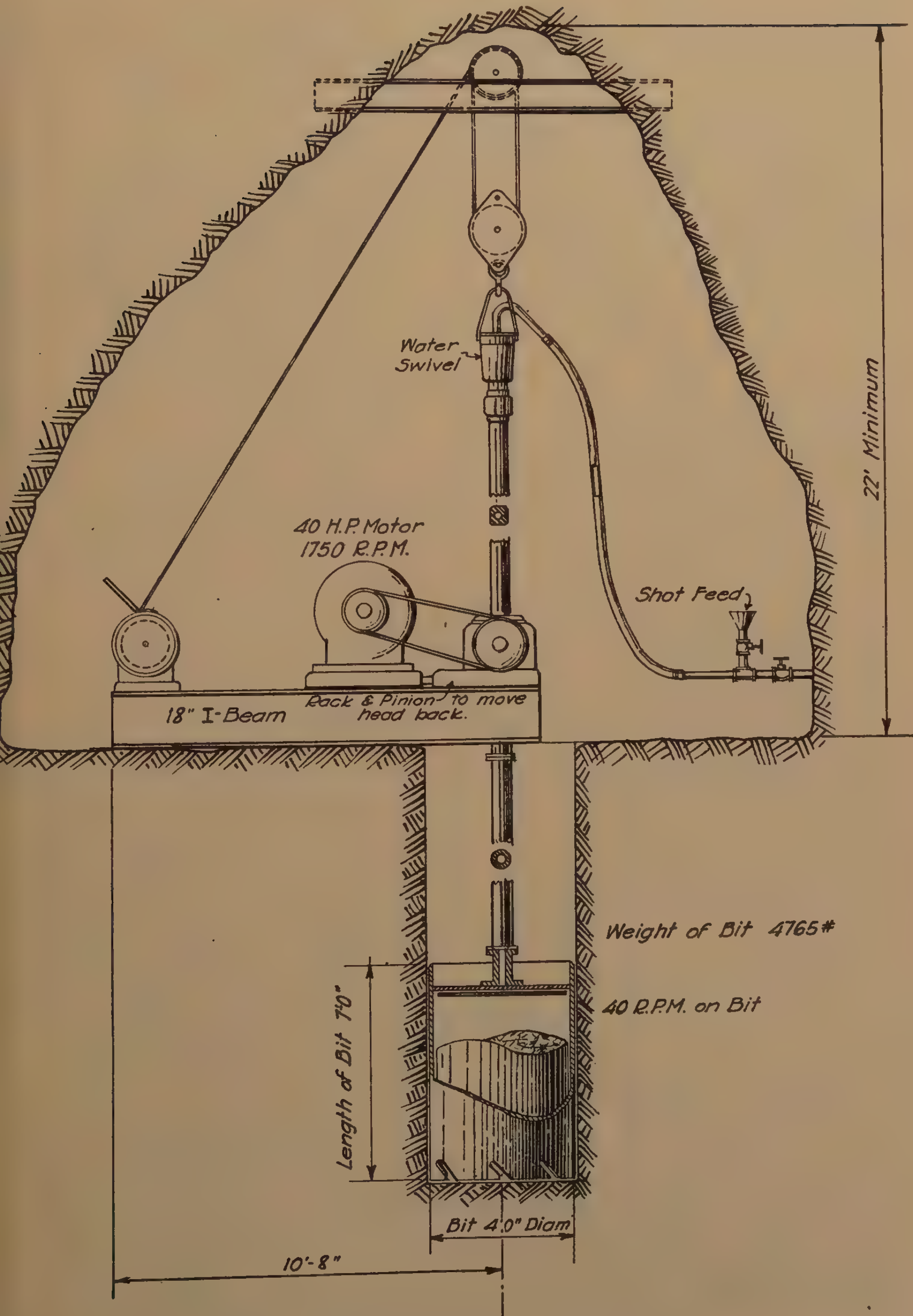


FIG. 10.—GENERALIZED SECTION OF CALYX DRILL.

they can be reversed when one end wears out. Wedge-type clamps are used for cable connections.

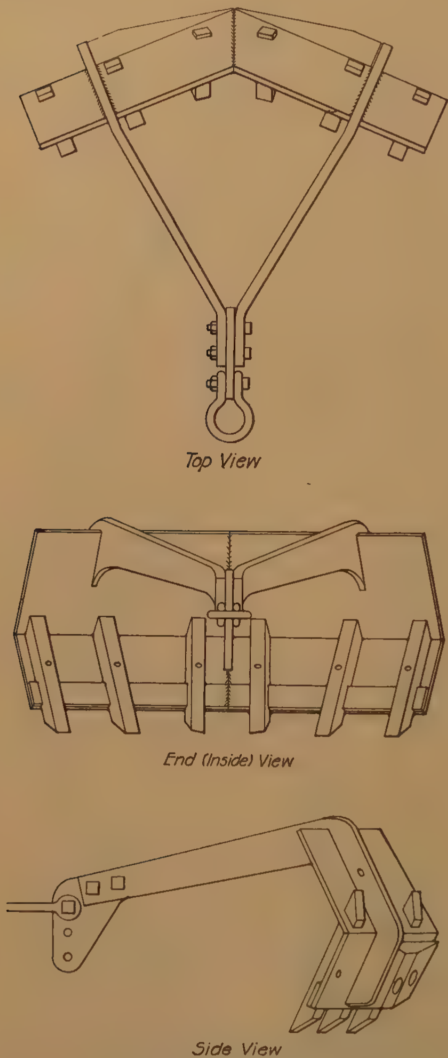


FIG. 11.—THE 48-INCH SCRAPER. OTHER SIZES ARE SIMILAR.

Steel-barrel Chute Liners.—Where it has been necessary to use old raises that had been carried up through old fill stopes, repair work has been a problem. After much experimentation a steel-barrel liner (Fig. 12) was designed and has been used with a surprising degree of success. This liner is made out of ¼-in. standard boiler plate.

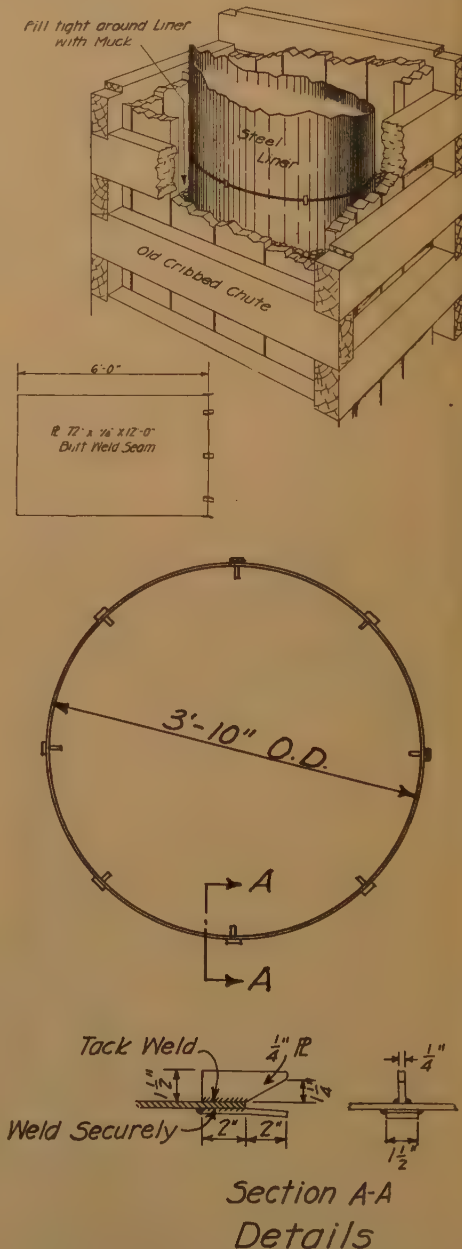


FIG. 12.—STEEL LINER FOR TIMBERED RAISES.

One plate will make one section without any waste of material. The plates are rolled in the form of a tube, 6 ft. long by 3 ft. 10 in. in diameter. The tubes are installed in the old raises and are held in place by simply filling around them with fine rock and small timber blocks. These tubes have been installed in raises ranging from 56 to 120 ft. in length and have handled as much as 15,000 tons of material per raise without being repaired.

Steel-rail Grizzlies for Stope Chutes.—Scrap steel ranging in weight from 60 to 90-lb. rails (depending on available material) is used exclusively for grizzlies (Fig. 13) in all stoping operations. Very little replacement has been necessary. The rails are moved from floor to floor as stoping operations progress. Special spacer blocks have been designed, which make the installation a very simple matter.

Acknowledgments

The writer wishes to express his appreciation to the engineering and operating staffs working under his supervision, for their aid in assembling these data.

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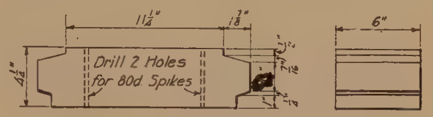
DISCUSSION

(*F. A. Wardlaw, Jr., presiding*)

McHenry Mosier,* Washington, D. C.—Are pillars simultaneously extracted on several levels of the mine? If so, what safety precautions are taken?

J. B. Pullen (author's reply).—The pillars are not mined simultaneously on several levels.

F. A. Wardlaw, Jr.,* Inspiration, Ariz.—In connection with your incline cut-and-fill method of stoping, do your miners not have



Detail Spacer Blocks for 60# Rail
(Vary for different weight Rails)

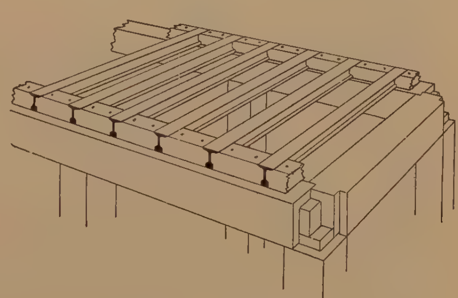


Fig. 13.—Stope grizzly.

trouble setting up on the incline muck pile to drill the back with drifter machines?

J. B. Pullen.—It is a little awkward at times, but they just level off a place and lay down lagging to set up on. There does not seem to be much trouble. Anyway, to more than offset this, there are the advantages of having the stope on a slope where both slushing the ore down and filling the stope with waste are helped by gravity.

C. F. Jackson,† Washington, D. C.—Do ground conditions at the United Verde make it possible sometimes to use a method similar to that often employed at Bisbee, there termed a semishrinkage method, whereby more than one cut of ore is taken before drawing the ore off and running in fill?

J. B. Pullen.—Ground conditions in many places would permit such an arrangement. This system is used to a limited extent.

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* Supervising Engineer, Metal Mining Research Section, Bureau of Mines.

* Assistant General Manager, Inspiration Consolidated Copper Company.

† Chief Engineer, Mining Division, Bureau of Mines.

Improvements in Mining Practice in the Butte District

By E. R. Borcherdt,* Member A.I.M.E.

(New York Meeting, February 1941)

Methods and equipment are subject to study in the Butte district at all times in order that advantage may be taken of any new developments that might serve to increase efficiency and lower costs.

Detailed time studies of each operation involved in the entire mining process have been of invaluable assistance in analyzing the individual operation for possible improvement. In other words, the fact that a practice has been in vogue for many years does not exempt it from change provided an easier and cheaper method can be found.

Exploration

Ore occurs in the Butte district in such manner that structures must be drifted on, no matter how small or unimportant they appear when intersected by a crosscut. Parallel structures also add to the crosscut requirements. This prospecting work, which formerly was done entirely by drifting and crosscutting, is now being supplemented by diamond drilling, to aid in greatly decreasing the amount of waste. In long-shot prospecting, or in second or third class prospecting, where the chances of developing commercial ore are not very great, diamond drilling is of particular value. In prospecting where chances of encountering commercial ore are certain or where crosscuts can be used later for haulage or ventilation, drilling obviously is not applicable.

Improvement in drilling machines and bits has contributed greatly to the present success of diamond drilling.

Manuscript received at the office of the Institute Jan. 16, 1941. Issued in Mining Technology, July 1941.

* Mines Research Engineer, Anaconda Copper Mining Co., Butte, Montana.

Stoping

For many years square-set filled stoping was the principal mining method used in the Butte district. Later timbered and untimbered rill stopes were used, also some horizontal cut-and-fill stoping where both ore and walls were particularly strong. Because of the amount of timber required for square-set stoping, and the relatively low production per stope unit, experiments with horizontal cut-and-fill stopes were begun in ore bodies of which many had fairly weak walls. It was felt that increase in the speed of the stoping operation, followed closely by filling, to avoid weight being produced by permitting ore and walls to remain insufficiently supported over comparatively long periods, would maintain the walls and backs in safe condition without the use of timber and without ore dilution from wall slough. Cycles of drilling, scraping ore, and filling were devised, which have materially increased stope production and decreased timber consumption, so that both stope labor and supply costs were lowered appreciably without materially affecting grade control.

In the larger mines of the district, where wide veins favor this stoping method, half of the total production is now being mined by horizontal cut-and-fill stopes (Fig. 1). The increase of the filling rate demanded by the increased stoping rate was the greatest obstacle encountered in the accelerated stoping operations. Finally this problem was solved by replacing the time-honored method of transferring filling waste by small cars, which were hoisted or lowered in the auxiliary shaft, by a transfer

system involving the use of a bottom-dump skip attached to the auxiliary hoist. The perfect functioning and simplicity of the bottom-dump skip as applied to the transfer pass, shown in Fig. 2, together with the use of large Granby cars for transporting waste to and from the pass system, has made it possible to gear the filling operation to the increased extraction speed-up. Fig. 3 shows the bottom-dump skip of approximately 4 tons capacity. The gross weight of the skip and load is less than the gross weight of a four-deck cage and four loaded cars containing 4800 lb. of waste. The time required for hoisting waste has been considerably reduced with this equipment, a saving that permits the auxiliary hoist to lower a larger proportion of the supplies and increases the time available for ore hoisting by the main hoist, which formerly was required to lower timber and other supplies.

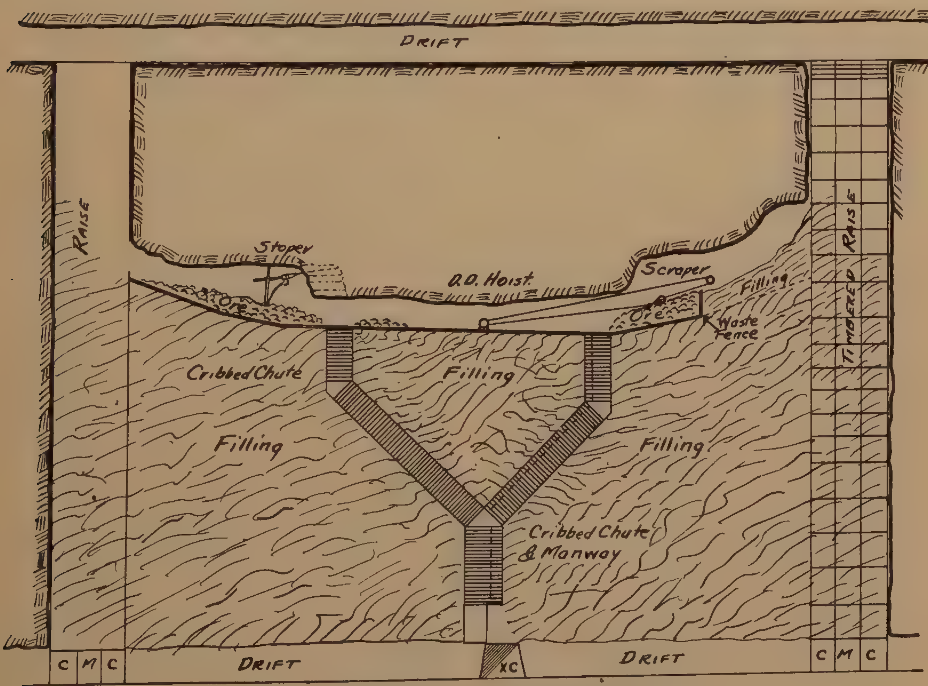
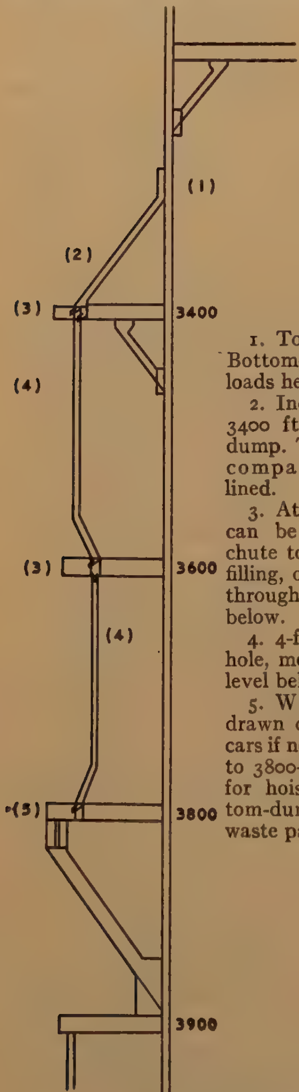


FIG. 1.—HORIZONTAL CUT-AND-FILL STOPE.

Fig. 4 shows the dumping mechanism, controlled by air cylinder, which operates at the top of the pass system as follows: As the engineer stops the bottom of the skip at a point above the yoke, the skip tender admits air to the cylinder, forcing the yoke into the shaft. As the engineer lowers the skip the door lugs engage the yoke, which retains the door in a stationary position and the dumping side of the skip is opened as it falls away from the door. The trajectory of the falling rock is sufficient to carry it over the bin lip without spillage into the shaft. The operation of these skips for waste transfer has been so satisfactory that similar skips are now being constructed to hoist 1000 tons of ore per day in balance from the 1000-ft. level to surface.

Greater flexibility permitted by the use of cribbed raise timber in adapting chute raising heights to variations in height of

the ore cuts has led to their adoption in place of the lined square-set stope chutes in general use over a period of many years. The cribbed chute is readily branched when



1. Top of waste pass. Bottom-dump skip unloads here.

2. Inclined raise from 3400 ft. to waste-skip dump. Timbered. Two-compartment. Rail-lined.

3. At all levels, waste can be drawn out of chute to cars, for stope filling, or can be passed through to skip pocket below.

4. 4-ft. Calyx drill hole, met by raise from level below.

5. Waste can be drawn out of chute to cars if needed, or passed to 3800-ft. skip pocket, for hoisting with bottom-dump skip to top of waste pass system 1.

FIG. 2.—WASTE TRANSFER SYSTEM EMPLOYING BOTTOM-DUMP SKIP IN AUXILIARY COMPARTMENT OF SHAFT.

more than one down chute in the stope is desirable.

Wet filling is being used successfully to afford maximum support of the hanging wall. About 10 to 15 per cent water is mixed with waste at the top or bottom of the fill raise, which is sufficient to cause it to run into hanging-wall voids otherwise impossible to fill effectively. The presence of clay from faults and alteration products of granite waste causes the material to consolidate, much like a weak concrete, after the water has drained from it. In addition to its compactness value in filling, it is an excellent firebreak, and also aids ventilation in preventing contamination by air movement through loosely consolidated gobs, a condition difficult to eliminate when timbered stopes are filled with dry waste.

The development of stope scraping practice and equipment has been an important adjunct to the success of the new stoping operation. Continuous rated, 15-hp. electric scraper hoists of compact design are standard equipment with 30-in. and 36-in. Holcomb scrapers. A reel (Fig. 5) holds sufficient rubber-covered cable to service the stope to completion and minimizes electrical labor required for cable extension.

New level development is entirely by footwall laterals with regularly spaced crosscuts for extraction. Ultimately, silling the full width of the ore on the level will be entirely eliminated, which should result in less cracking and weakening of both ore and walls above these sills.

Economies resulting from increased unit stope production will be further emphasized by permitting concentration of stoping areas, which will make possible better supervision, reduction in repairs due to lessened need for maintenance of sill footage, and greater efficiency in ventilation of a more concentrated working area.

HAULAGE

Small 18-in. gauge end-dump or side-dump mine cars of about 14 cu. ft. capacity are used because in the early days cage hoisting was done through small shafts. About 8000 of these cars were in service in the various mines, and while the numerous

advantages obtainable with the use of larger cars were recognized, it was not until 1937 that large Granby-type cars were adopted in the larger mines.

The Granby cars (Fig. 6), which were

was excessive when either spring or rubber mountings were carried on the inside of the wheels. Satisfactory air-operated, retractable, Granby-car dumps have been developed.

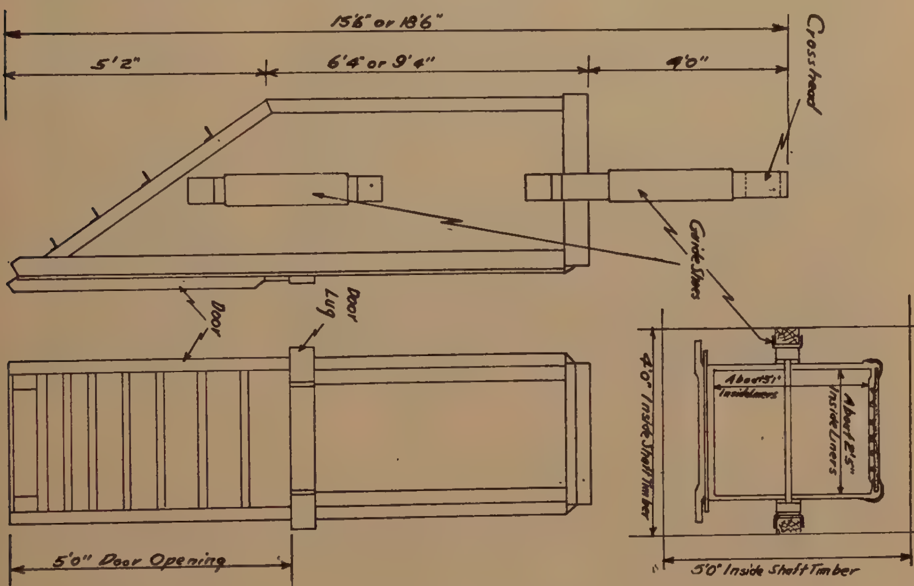


FIG. 3.—BOTTOM-DUMP SKIP.

designed and built by a well-known car-builder, are 9 ft. 6 in. over-all length, and are equipped with ½-size MCB couplers, spring draft gear, and 14-in. steel wheels. In the interest of weight reduction and corrosion resistance the bodies and frames were fabricated from high-strength copper alloy sheets and structurals. The body capacity is 53 cu. ft., level with the ends, with an inside body length of 7 ft., which gives a loading capacity of between 3 and 3½ tons, or equal to the capacity of between five and six of the smaller cars. Height and width of body dimensions adopted were the maximum that could be lowered through the shaft compartments. The over-all weight of the car is approximately 3500 pounds.

The application of a car of this size to 18-in. gauge required development of an outboard spring construction, as side sway

The use of large cars accentuated the need for better track installations. A change from 25-lb. rail to 40-lb. rail, with minimum curves of 18-ft. radius and latch switches with curved points, was made. Reduced friction obtained with the large cars in hauling the same ore load with one-fifth of the wheels required with the smaller cars permitted a decrease in track grades.

Charging equipment for extra batteries has been installed, to reduce the number of locomotives required on each level and to ensure availability of properly charged batteries at the beginning of each shift. Experience with air or fluid-operated brakes suggests a marked reduction in battery drain resulting from spotting of cars at chutes with both power and brakes applied.

The adoption of the new haulage equipment has effectively reduced the number of

locomotive crews required, and has decreased delays in servicing mechanical loader crews, so that greater advantage can be taken of the high potential capacity of mechanical loaders. A substantial reduction in the underground haulage cost per ton of material handled has been achieved with the larger car equipment.

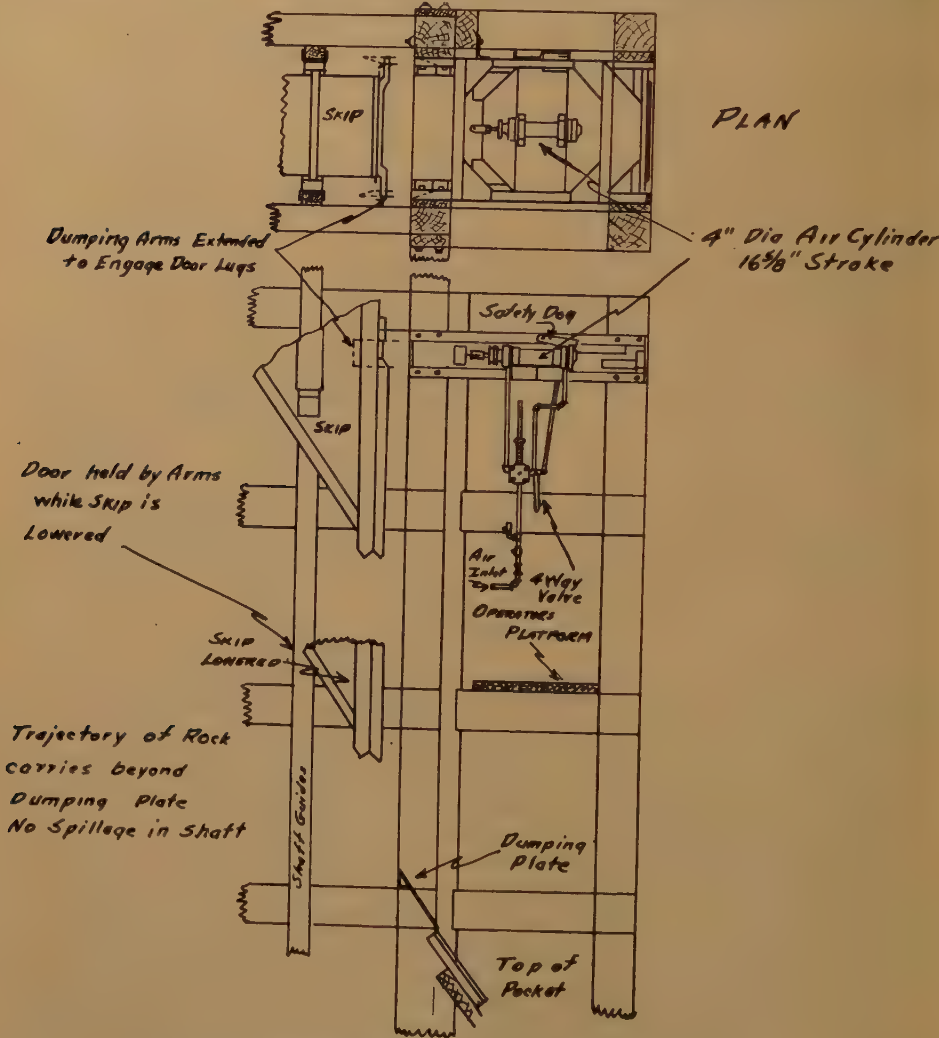


FIG. 4.—DUMPING MECHANISM OPERATED BY AIR CYLINDER.

DRILLING AND BLASTING

Rock-drill requirements per unit of daily ore production have been gradually reduced through improved mine organization and improved drill design. The proportion of self-rotated stopers to drifters in use is about 4 to 1, while one jackhammer is used for every 10 self-rotated stopers. Automatic feed, power feed, and hand-cranked drifters are employed; the latter where drilling conditions make it impossible to take advantage of the higher over-all drilling speed of automatic feeds.

Small jumbos, carrying either two or four drifter drills, are used where increased

speed of advance is required. Reverse-feed stopers mounted on pneumatic bars are being used successfully for drilling flat

FIG. 5.

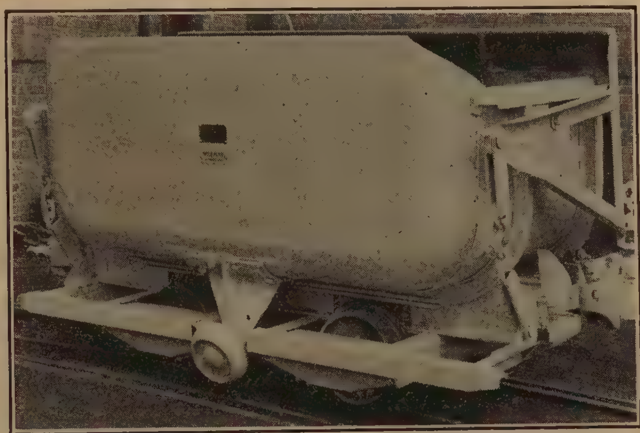


FIG. 6.

FIG. 5.—ELECTRIC CABLE REEL FOR SCRAPER HOISTS USED IN STOPES.
FIG. 6.—GRANBY CAR.

holes in horizontal cut-and-fill stopes. The flat-hole drilling gives a more regular stope back than that obtained with the use of self-rotated stopers, which is desirable from a safety standpoint. The use of jack-hammers mounted on a "Mexican set-up" is being expanded for drilling flat holes in soft-ground stopes.

The use of detachable drill bits, which has been standard practice in this district for many years, was changed, several years ago, from the Hawkesworth tongue-and-groove bit to a well-known threaded bit. High-speed, fully automatic grinding machines, which were developed for sharpening the Hawkesworth bit, have been adapted to the sharpening of screw bits. We feel that the advantage in higher drilling speed, reduced steel breakage, and reduced rock-drill maintenance obtained by using bit gauges as small as possible greatly offsets the advantage of a greater number of reground bits obtainable with the use of large-diameter starter bits. Experimental work is being carried on to investigate the possibilities of further reducing bit size by substituting 1-in. Q.O. drill rods with ⅞-in. Q.O.

Numerous tests of alloy-steel drill rods have thus far failed to show an all-round advantage over straight-carbon drill steel.

The use of Gelex, which tests in Butte ground ascertained to have approximately the same execution value per stick as that of 40 per cent gelatin, but with the advantage of 16 per cent increase in stick count per case, has been in general use since 1932. The use of bunch blasting,* which was developed to provide fuse-spitting means when carbide lamps were discontinued, continues to be an important aid in the reduction of blasting accidents. The value of space blasting has been definitely determined. An improvement has been made in bench-type cap crimpers previously used, whereby a novel form of crimp is sufficiently watertight to make the use of waterproofing compounds unnecessary. Machine-loaded clay tamping in red paraffined cartridges has been in general use for some time, but the recent development of a fire-proof cotton-asbestos tamping material of stemming value comparable to that of clay,

* J. J. Carrigan: Anaconda Method of Bunch Blasting. *Min. and Met.* (Aug. 1936) 384.

but with the added advantage of light weight and low cost, will no doubt replace clay.

is being expanded. Maximum advantages in increasing speed of driving and reducing costs are obtained where conditions of tem-

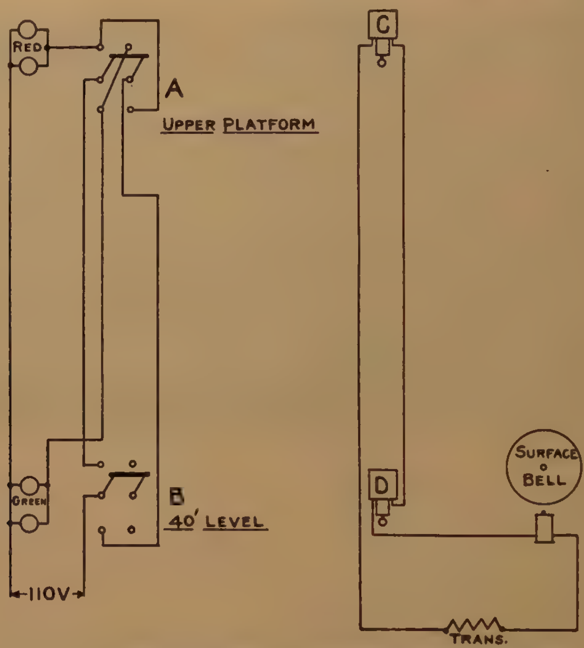


FIG. 7.—ELECTRIC SIGNAL SYSTEM FOR DOUBLE LOADING.

Starting with switch A in up position, man loading lower deck puts switch B in tip position, which lights the red lights on upper platform and indicates that men may be loaded on upper deck. As soon as men are on cage, man in charge of upper loading platform pulls down pull-switch C and throws down switch A. This puts out red lights and lights green lights on 40-ft. level, indicating that cages may be lowered. As soon as cage goes down man on upper platform releases pull-switch C and puts D out of action. When upper platform is not in use, C is wedged in closed position and D used as usual.

For switches A and B, use G.E. 2897-39 in Crouse-Hinds Condulet FS2 with DS128 cover.

perature and humidity are severe. Continued improvements in design have resulted in more massive construction, with resulting decrease in maintenance costs. The greater headroom and cross section required for the operation of the larger loaders is a distinct disadvantage when driving in waste. In tunnel work, where efforts are concentrated on the rapid advance of one heading, a loader of this size may be justified, but in mine work its loading capacity is far too great for the number of cars that can be provided in most mines.

TIMBER

Consumption of timber has been reduced gradually from 24 board feet per ton of ore in 1925 to 14 board feet per ton at the present time. This reduction has been accomplished by standardization, centralization of cutting into "ready to use" material at the saw mills, application of mining methods requiring less timber, and increased use of salvaged timber.

MECHANICAL LOADING

The field of mechanical loaders such as the Eimco-Finlay and the Gardner-Denver

Calyx Drilling

For a number of years, underground Calyx drilling of 36-in. holes has been done in the Butte district as an aid to ventilation of bottom-level development work. Recently equipment for the drilling of 48-in. holes has been purchased to enlarge further the scope of Calyx holes for use as ore and waste passes.

8 9 10

Fig. 8.—"Butting-in" saws used for rip cuts in framing shaft timber.
Fig. 9.—Portable electric saw.
Note rip cuts in end plate, which were made on "butting in" saws.
Fig. 10.—Another type of electric saw, equipped with ¾-inch dado cutter.
This is used instead of axe and chisel, for cleaning out daps.

Mine Piping

Tests of galvanized-iron piping for underground air and water lines over a number of years have demonstrated its value in effectively reducing surface corrosion and pitting of such lines. The use of Victaulic couplings for both air and water lines is increasing. The advantages of these couplings over threaded couplings lies in the ease of making a leakproof connection and the ease with which pipe may be salvaged from abandoned drifts and raises.

Skips and Cages

The weights of skips and cages have been reduced by the use of high-strength copper-alloy steel, as reduction in weight was necessitated by the deepening of shafts. Rubber linings in skip bottoms have continued to be satisfactory in reducing the

amount of material that adheres to the skip bottoms. The effect of rope stretch upon skip loading at bottom-level pockets is serious, in that it results in greatly increased spill. The use of deflectors to reduce this amount of spill has met with some success. The simultaneous loading of men on alternate decks of cages has been applied successfully to the hoisting and lowering of the shifts at large mines in the district. The electric signal hook-up, which assures against movement of the cages before the loading or unloading of both decks is completed, is shown in Fig. 7.

FIG. 11.—BULLETIN BOARD.

Radio signals of the Magma type are used in main upcast air shafts where periodic inspection and repair require hoisting and lowering of repairmen. The need for conventional electric signal wiring, of which the wet corrosive atmosphere is extremely destructive, has been eliminated by the radio signal attached to the cage.

Disposal of Surface Waste

For many years excess waste was hoisted by skip or cars and transferred to a side-dump larry car, from which it was dumped from a timber trestle. The cost of trestle construction was high, therefore trucks were substituted for the cars, as they do not require a trestle.

Timber Framing

The adoption of a tunnel set to give increased strength over the stepdown framing formerly used in drift and crosscut support, has required design of an automatic framing machine, which also is adapted to the framing of cribbed raise timber. The framing of shaft timber, which for many years was done by hand at the various mine carpenter shops, has now been centralized at the Rocker framing plant, where mechanical equipment is available, and this has materially decreased the cost of shaft-timber framing. Figs. 8, 9 and 10 show the equipment used for this work.

Safety

The principal additions to our Butte mines safety program for the past several years are:

1. Foremanship training through monthly get-together dinners for all bosses.

2. Safety education through daily morning meetings of mine crews at which safety talks, illustrated, are given, or motion pictures of safe methods are shown.

3. Maintenance of interest in safety by monthly cash prize awards. A ticket is given each man who works 18 or more shifts during the month and does not suffer a lost-time injury. These tickets are shaken up in a box and one or more of them are drawn for the $25 prize. Mines employing fewer than 300 men have one prize of $25, those employing 300 to 500 men have two $25 prizes, mines of 600 to 900 men have three $25 prizes and mines over 900 men have four $25 prizes. There is also a $25 prize for surface laborers and another for the craftsmen, as well as one $10 prize for men of nonoperating mines.

4. Mine-safety bulletin boards have been enlarged and elaborated (Fig. 11).

5. Items of interest and lessons in safety, particularly descriptions of actual accidents, are sent to each shift boss in the form of information bulletins twice each week.

DISCUSSION

T. T. Read,* New York, N. Y.—This extremely important paper is especially interesting to me because I spent some time at Butte, last summer, seeing the developments that have been so tersely described by Mr. Borcherdt. Perhaps some observations from the viewpoint of a visitor who first saw the district 30 years ago may help readers to grasp the full import of what has here been so briefly described.

Much has been published on the geology of the Butte ore deposits, ranging from the folio of the U. S. Geological Survey that appeared in 1897 to Grout and Balk's detailed study of internal structures in the Boulder batholith, published by the Geological Society of America in 1934, but for most people the three papers on the subject by Reno H. Sales, F. A. Linforth, D. C. Bard, and M. H. Gidel in Vol. 46 (Montana volume) of the Transactions will be most convenient for consultation.

For the visiting mining engineer, it is perhaps enough to visualize a batholith cut by an intricate system of ore-bearing veins that may be divided roughly into three main systems, with general east-west, northwest-southeast, and northeast-southwest strikes. These are cut by later fault fissures, with displacements ranging from a few feet up to 300; creating mining problems of great interest as well as technical difficulty, since the thickness of the veins varies in both the horizontal and vertical planes. Ore shoots are as much as 2000 ft. long and many important ore bodies appear only in depth. There is a manganese zone of mineralization, a zinc zone and a copper zone, the latter being the deep one. In the early days the ownership of these deposits was in the hands of a large number of separate companies, which worked them through many small shafts. The litigation that developed as a result of the law of the apex is worth mentioning only to note that J. Parke Channing said, in 1903, that it had "forced development to such excess that certain drifts will have to be retimbered four times before the ore is stoped." Thus man-made as well as natural conditions contributed to the complexity with which present technology must deal. Successive consolidations reduced the number of separate operations, and the paper on Mining Methods in the Butte District, which appeared in Vol. 72 of the Transactions (1923) was the joint product of over a dozen engineers of six different companies. Now there is only one operating company, though others still own property.

While it is practicable to consolidate ownership of such a district, the consolidation of operations is much more difficult, for the many working shafts inherited from an earlier day bore some relation to the natural complexity of the vein structure. By 1923 a rough standardization on rectangular four-compartment shafts, 20 by 7 ft. in outside dimensions, had been arrived at. Originally hoisting was done in small cars on multideck cages; Mr. Borcherdt gives data that apply to the substitution of skip-hoisting, and it is clear why it is not practicable to use the large skips that have been adopted elsewhere for use in large shafts. The hoisting engines originally were steam-operated, but were changed to compressed air some 30 years ago. Gradually the compressed-air hoists were displaced by electric hoists, but the large central air-compressing station, op-

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erated by synchronous motors, now functions as a central air supply for drilling. One mine, the Emma, owned by the Butte Copper and Zinc Co. and operated by Anaconda under lease as a source of zinc and manganese ore, still uses its flat-rope steam hoist and sends out its ore by truck, because it has no railroad connection.

In the 1923 paper on Butte mining methods it was said that "fully 80 per cent of the ore" was then being produced by square-set mining. Mr. Borcherdt says that in the larger mines, where conditions are favorable, half of the ore is now being produced by horizontal cut and fill. A visitor might guess that the continued application of the intelligent and effective study that has been given to the mining problem may lead eventually to production of 80 per cent of the ore by cut and fill, with the resultant lowering of operating cost suggested by the paper.

The copper content of the ore produced by Anaconda has been fairly uniform over a long period of years, study of the published records indicating that it has ranged from a low of 80 lb. to a high of 120 lb. copper per ton. This is not surprising, since mining operations have long been controlled by careful sampling and geological study, as described by Daly and Linforth in their paper in Vol. 68 of the TRANSACTIONS (1923).

The brief description of drilling practice is a modest presentation of the conclusions so far derived from a large amount of careful study. The detachable bits now used do not differ greatly in design from the standard jackbit, tests of the type of bits developed for faster drilling in the Tri-State district, as described by Nicolson [*Trans.* A.I.M.E. (1940) **141**, 64] having shown that under the conditions at Butte they did not give as good results as the more conventional pattern. A central plant sharpens dulled bits by grinding on automatic grinders. Each mine orders its daily supply of sharp bits, being charged for them and credited with the dulled bits returned; the charges being so adjusted that the bit plant makes neither a profit nor a loss. Broken bits and scrap are sold to the company smelter, which casts them into balls for ball mills, so there is no waste of steel except through abrasion in use. A more detailed paper on drilling practices would be of great interest, as would be one on the special problems involved in the use of mechanical loaders under Butte conditions. The brief reference to the use of stemming made from noncombustible cotton waste is the first report on its use that I have seen in print. I understand it has been tried in a number of places, and private advice from another mining company says that the men like it because it does not run out of an upper hole as sand stemming does when a cartridge breaks; also, it is said to save enough powder to justify its use. However, it is not easy to induce contractors to use less powder in a hole than they have been using. It is to be hoped that we may hear from a number of operating engineers as to their actual experience with it.

The very brief reference made to the use of Calyx drilling in development work is supplemented by Mr. Pullen's detailed description of its use at United Verde (this volume, page 36). It should be noted for the record, however, that the first use of a Calyx drill for underground drilling was at Butte. The timber framing plant at Rocker was described in detail in the Montana volume already referred to, and about the only differences to be noted are the centralization of shaft timbering there, as noted by Mr. Borcherdt, and the provision of a treating plant using an arsenical solution. This plant treats ties for the Northern Pacific Railroad as well as such mine timber as justifies its cost.

The brief paragraph on p. 44 on footwall development passes lightly over what is really a major change in the mining system. Formerly levels were established, drifts driven on the veins and the whole width of the vein silled out before stoping was begun, with the result that it was difficult and expensive to maintain the timber on the level while using it for haulage purposes. More complete knowledge of underground geology and more extensive use of diamond drilling instead of drifts for exploratory work makes it possible, in general, to keep the haulage levels in the footwall, the cost of crosscuts being less than that resulting from the attempt to do without them. With increasing pressures in increasing depth, this will become correspondingly more important. In this, as in the rest of the paper, the results of careful study have been presented with great brevity and equally great modesty, and it is to be hoped they will receive the credit they so richly deserve.

Extending the Scope of Placer Dredging

By C. M. Romanowitz,* Member, and H. A. Sawin,* Associate Member A.I.M.E.

(New York Meeting, February 1941)

Placer dredging as we know it today, especially gold dredging, is an industry about 40 years old, dating from the beginning of this century, when a few mining men in California saw the possibilities in adapting earlier dredging methods to California placer fields in and immediately adjacent to streams. At first it was thought that dredges must be operated in existing stream beds, but experience soon pointed the way to dredging alluvial deposits of old stream beds, operating on landlocked dredging ponds.

Within the first five or six years of modern dredging, experience was gained so rapidly that dredges to dig 60 ft. below water level were designed and built. This depth was believed by many operators to be the maximum at which successful dredging could be carried on. Alluvial deposits of gold, suitable for dredging, were readily handled by equipment available and developed early in the industry's history. Much of the gravel was easily dug and if deposits were cemented, contained large boulders, or for some other reason could not be handled by a dredge, the ground was abandoned in favor of other deposits that could be handled easily. Many failures, adversely affecting the dredging industry, resulted from attempts at that time to extend the scope of placer dredging with equipment not suited to the formation being dredged. Dredging ground suited to early equipment was exhausted eventually, but many supposedly undredgeable alluvial fields containing gold in commercial quantities were found and became known to dredgemen.

Improved equipment was needed to work deposits known to exist at great depths or associated with unusual obstructions. Operating companies looked to dredge manufacturers and metallurgists, particularly manganese-steel foundries, for improved designs and materials. Experience developed in California has spread throughout the world, and dredges of improved types have been adapted to the mining of platinum and tin as well as gold.

Space will not permit a description of all the improvements in dredging since the early days, but a few of the recent developments are worthy of note, because they make possible the dredging of placer deposits of great depth, those having hard, irregular bedrock or containing large boulders in great quantities, and compacted or cemented gravel deposits.

Deep Dredging

At Hammonton, Calif., Yuba Consolidated Gold Fields has, within the past two years, built its dredge No. 20. The company has operated continuously in that field since 1904, digging successively at maximum depths of 60 ft., 80 ft., 112 ft., and now 124 ft. below water level. The first dredges were equipped with buckets of 6 and 7½-cu. ft. capacity; six dredges operating in the field today have 18-cu. ft. buckets, digging now 65 to 124 ft. below water level.

Some ground near Hammonton has been dug for the third time, owing to the devel-

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* Yuba Manufacturing Co., San Francisco, California.

opment of dredges and equipment to operate at increased digging depths. Yuba No. 20, capable of digging 124 ft. below water level, can operate against a bank 50 ft. high, or more, making a possible digging depth of about 174 ft. below ground level. Yuba No. 17, another deep-digging dredge, is capable of digging 112 ft. below water level against a bank of 50 ft., or 162 ft. below ground level. Both Nos. 17 and 20 are equipped with Perry bucket idlers and Yuba mud-pumping systems. These devices have been described previously, but the continued experience with them proves that deep dredging can be carried on successfully and in deposits of comparatively low value when such aids are used. A Perry idler, mounted on the underside of the digging ladder about midway, controls the bucket-line catenary, contributes greatly to the proper functioning of the line through better distribution of weight, and is directly responsible for reducing wear on pins, bushings, tumbler plates and buckets.

Yuba No. 20's main drive, based on experience with Capital No. 4, an 18-cu. ft. dredge digging 82 ft. below water level, is powered by two 300-hp. alternating-current motors. These motors are mounted just aft of the upper tumbler and are connected by V-belts to the pulley shaft. These a.c. motors are synchronized electrically and mechanically, have proved to be efficient, and provide flexibility that was not available in the older system of main drives. On dredges of smaller bucket capacity, only one main drive motor is used with V-belts, and the same good results are obtained.

Removal of Fines

The need for removing the fines, composed of fine sand and silt, from the deep ponds of Hammonton became apparent almost immediately after No. 17 reached maximum depth. One handicap to deep dredging to which necessity forced an answer is the tendency of fines to slide forward in the dredge pond, where they accumulate on the bottom to depths of 30 ft. or more. This condition develops because deep-digging dredges make a wide face or total width of cut necessary, which, together with the great depth, produces a larger quantity of fines than ordinarily is experienced in dredging. The comparatively long time required for the dredge to complete its work in one position in the pond leaves the opposite side of the pond in an undisturbed condition for a long period. This permits the fines to settle in the large quantities first noticed in Yuba No. 17's pond, and creates a serious problem. These fines contain no metal, obstruct side movement of the digging ladder and buckets, and thus prevent free swinging of the dredge. Unless removed, the fines would be redredged time after time, to no purpose, and would accumulate to a point where dredging would be stopped or seriously impaired. Some deep-digging dredges, not equipped for removing such fines, are operating under a great handicap. The Yuba mud-pumping system was developed for the express purpose of removing these fines from the forward end of the ponds of deep-digging dredges. The fines are removed through a suction pipe line carried in the ladder, and discharged overboard through a pipe line carried up the stacker, or through a floating pipe line across the pond to a point sufficiently removed from the dredge to be kept permanently out of the pond. Tests indicate that no metal is found in these fines, but should any be present, having dropped from the ladder or fallen from the bank face, it would settle to the bottom of the fines and be recovered by the buckets, as it is customary to pump only about 20 ft. from the top.

New Gold-saving Tables

Another departure first used on Yuba No. 20, which increases the efficiency of dredges, is the new Yuba table system. Just prior to the building of Yuba No. 20,

a gold-saving test, conducted over a long period of time, was completed, and indicated that the standard arrangement of tables could be greatly improved. This new system calls for a design and arrangement of tables based on the total width of sluices rather than on a basis of total area as previously used. On the previously used basis, Yuba No. 20 would require 8400 sq. ft. of table area, but the gold-saving tables on this dredge have only 5600 sq. ft. of riffled table area. These tables proved to be more efficient than tables on other dredges of the same bucket capacity operating in the same territory and handling similar formations. The previously used system of table arrangement called for riffles on both athwartship and longitudinal tables. On the athwartship tables, water could be distributed evenly to produce good gold-saving conditions. In the longitudinal tables, fewer in number than the athwartship tables, and each receiving fines and water from two or more athwartship tables, the volume increased to a point where the gold-saving efficiency was seriously impaired. It was found that a large percentage of the gold caught was on the forward tables in a small triangular area. Tests proved that the fine gold delivered to the longitudinal sluices did not have sufficient chance to come in contact with the mercury or to be caught by other means, because of the large volume of water.

The "total width of sluice" system calls for all riffled area to be in athwartship sluices. These sluices extend as far aft on the dredge as required to properly treat material received from the screen. They all discharge into two longitudinal sluices, one outboard on each side of the dredge. Longitudinal sluices are not equipped with riffles, and no attempt is made to save gold because of the large volume of sand tailings and water being carried overboard. Tests have shown that riffles are not needed in the longitudinal sluices because of the high efficiency of the athwartship sluices. When required, tables can be double-decked and the Yuba system is so arranged that the gold-bearing material can be distributed from any given point to any other desired sluice below that point. In actual practice, the results show that gold caught on these tables is distributed in a narrow strip parallel to the screen, extending aft toward the lower end of the system. Fine gold has a better chance to be caught, as the water can be regulated properly to each sluice, and can be varied in volume to suit requirements. This is one important step toward increasing the efficiency of placer dredges.

Length of Hull

The success with deep-digging dredges is dependent also on consideration given to the length of hull. The hulls of Yuba's No. 17 and No. 20 are longer than is required for flotation and trim only, owing to the necessity for keeping tailings from sliding into the digging position when the bucket line is working on or near bedrock. To avoid this, the hulls are so long that ballast must be used to correct the trim and offset the known excess flotation. Yuba No. 20's hull is not as deep as that of Yuba No. 17, but the length was so increased that it was necessary to install an auxiliary stacker between the screen and main stacker, and a well hole was also placed at the stern of the dredge. The additional stacker equipment needed because of the unusually long hull provides an arrangement at the stern that is admired by all dredgemen visiting these operations. The operators themselves state that the arrangement at the stern of Yuba No. 20 makes the best working conditions possible. The tailings mentioned here are not to be confused with the fines referred to previously, which are removed by the Yuba mud-pumping system. If proper provision were not made to keep these tailings away from the digging operations, the result would be as serious, perhaps more

serious, than that of the fines mentioned earlier.

Buckets and Other Improved Equipment

Deep digging is leading to the development of many placer fields, and dredging properties having a depth of more than 174 ft. below ground level are being considered. The use of larger buckets in connection with these deep-digging dredges is also being studied, with increased capacity in view. Deep-dredging costs today show that in similar formations and when compared with shallower operations, an increased operating cost of approximately one cent per cubic yard should be taken into consideration when studying a proposed deep-digging dredging area. If formations at the greater depths increase in hardness, as has been found in some places, the operating costs then increase more than that amount.

Dredging fields in northern California, including those of Trinity and Siskiyou Counties, are well known for conditions that proved insurmountable to early dredges built from experience gained in dredging ordinary gravel deposits. These fields are noted for conditions generally unfavorable for dredging, particularly because of large boulders, cemented gravel, and hard uneven bedrock. To dredge deposits that have been known for many years, and on which early-day dredges had been abandoned, two modern dredges have been built recently. One is operated by Yuba Consolidated Gold Fields in Siskiyou County, near Callahan, and the other by Carrville Gold Co. in Trinity County.

Both dredges are equipped with oversize main drive units which in ordinary circumstances would be sufficient for driving an 18-cu. ft. bucket line. The Callahan dredge has 9-cu. ft. buckets to dig 35 ft. below water level, and is so arranged that the hull and digging ladder can be extended to dredge deeper ground if necessary.

The Carrville dredge has 12-cu. ft. buckets to dig 50 ft. below water level and can readily be adapted at a later date to dig 75 ft. Both dredges have standard-type structural steel hulls and superstructure, with spuds designed for exceptionally hard digging. The digging units and spuds are far heavier than would be usual with buckets of the sizes used. The buckets themselves are of an improved design and much heavier than normal 9 and 12-cu. ft. buckets. This extra strength and sufficient main drive power on these two dredges have been greatly responsible for the successful dredging of their properties. As an indication of the hard digging accomplished by the Carrville dredge, it is interesting to note that five sets of manganese-steel bucket lips were installed during the first year of operation. At times the ground dug is so hard that chips of manganese steel are torn from the lips and are found on the gold-saving tables. To our knowledge, this is the only dredge where such severe wear occurs and the dredge continues in operation.

The lower tumbler on the Carrville dredge is a duplicate of those used on 18-cu. ft. dredges and is interchangeable with dredges of that size used in California today. The Callahan lower tumbler, while not identical, is equal in size and strength. Lower tumbler bearings equipped with self-centering hardened-steel thrust plates, developed by practical dredgemen, are used, and no trouble has been experienced even after many months of hard digging. This type of thrust is used on many dredges today.

The digging ladders on both of these dredges are of the plate-girder design of extra sturdy construction, having the conventional ladder pan for returning the droppings from the buckets to the lower end of the ladder. On the Carrville dredge, the ladder was first equipped with 1-in. wearing plates on the bottom, but these were soon replaced by three wide bars 3 in. thick. These in turn were replaced by heavy cast-

ings about 7 in. thick. These heavy wearing parts are necessary because in tough formation, such as cemented gravel, it is customary to allow the bucket-ladder to ride on the buckets digging on the underside of the ladder. This practice, possible only with sturdy dredge construction, helps the Carrville dredge to overcome the difficult formation that shortens the life of the lips.

Handling Large Boulders

Large boulders have always been a source of trouble for dredge operators. They have been directly responsible for abandonment of operations on properties in northern California, Idaho and elsewhere, and have been the principal reason for not attempting work on many properties. As dredging ground becomes more and more difficult to find, and with the inducement of thirty-five-dollar gold, dredge operators look to manufacturers to supply units designed to overcome boulders and other unusual dredging conditions.

Among new developments brought about by the widening scope of dredging are buckets of relatively long pitch, flattened bowl-shaped bodies, and greatly improved lip design. When dredging in boulder formation using standard buckets, serious difficulties are encountered as the buckets straighten out when leaving the lower tumbler. Large boulders are cramped between the buckets, causing the lips to be torn off or damaged, the hoods to be torn or crushed, or other difficulties. Until recently, the method employed to overcome this condition was to use an open link or tray between buckets. This system caused uneven digging because of the variation in the line of travel of buckets separated by the links or trays. Many operators today have redesigned their buckets along the lines of the buckets mentioned here. This new type of bucket was designed to overcome these difficulties and has fulfilled all expectations, thus helping to extend the scope of placer dredging. The 12-cu. ft. buckets on the Carrville dredge have a pitch of 41 in., which is as long as the normal 18-cu. ft. bucket and longer than the standard 13½-cu. ft. bucket, which has a pitch of 38 in. for normal dredging conditions. During two years of operations, no boulders have been wedged between the buckets on the Carrville dredge, although many large boulders have been dredged.

At Carrville a rock eliminator is used at the main drive, enabling the operator to pass overboard at the side of the dredge boulders that should not be dropped into the screen. This is not a new departure in dredging, but it is important to provide such equipment where large boulders are to be handled.

Porter and Company is operating a small dredge near Granite City, Oregon, which also is dredging successfully in deposits long known for bad boulder conditions. Other operators abandoned these areas, principally because two standard-type bucket dredges were unsuccessful. Lacking sufficient power, and not being equipped properly otherwise, they could not handle the gravel, which includes large boulders, nor meet other severe conditions. One of these dredges was moved to a property in Idaho, and under normal digging conditions has done good work. Porter and Company's dredge is performing beyond all expectations, handling its ground economically. The buckets on this dredge are 4¼ cu. ft. in capacity and of the special design referred to above, having a pitch of 29 in., which is several inches longer than would be required for ordinary digging. The dredge throughout is of heavy design.

Another recently started operation, which promises to establish a precedent for equipment to be used on similar ground under difficult conditions, is that of Supia Gold Dredging Company's dredge on the Supia River, in Colombia, South America. The property is shallow and was known to contain many large boulders. A study of

the conditions indicated the need for radical changes from standard dredge design. This operation is under the direction of F. W. Griffin, with A. C. Clements as superintendent. The departures from standard practice in dredge design were acceptable to them, and operations to date have proved that proper equipment was employed. The shallow ground did not permit use of a rock eliminator like that used on the Carrville dredge, and the boulders were so numerous that they could not be dumped in the pond just aft of the stern of the dredge. To provide for these two limitations, the dredge has 8-cu. ft. buckets of sturdy design and long pitch. The boulders dredged are dumped into the screen, which is made up of two layers of plate to provide for excessive load and wear. The outside perforated plates are of mild steel, ½ in. thick. The inside renewable perforated plates are steel, ¾ in. thick, making a total thickness of 1¼ in. in the screening area. Large boulders pass from the screen directly onto a specially constructed stacker having a belt 48 in. wide and idlers closely spaced.

Boulders have been larger and more numerous than was at first expected, and it has been necessary to take some out of the buckets at the lower tumbler. Other boulders, too large to bring to the surface, have been buried. Boulders that cannot be brought to the surface cause the greatest delay, because of the comparatively long time required to dig a hole and bury them, so that the dredge may move forward.

Hulls of dredges shipped out of the United States cannot be of the portable pontoon type unless an excessively high freight charge is paid. For Supia a special type of hull was designed to avoid measurement penalties and to provide one that could be readily assembled in the field.

Conclusion

The managements of the properties referred to are entitled to much credit in handling the difficult problems. Management is largely responsible for the successful use of proper equipment on a placer-dredging operation.

This paper is necessarily limited to the discussion of bucket-line dredges and reference to the development of so-called "doodle bugs" and other washing-plant equipment for operating on small deposits has been avoided. These have been covered quite thoroughly by other writers and, in themselves, represent another picture of the widened scope of dredging.* The intent of this paper is not so much to point out particular improvements on certain dredges but to indicate the trend toward placer-dredge design that incorporates features and equipment needed to operate successfully under conditions that not so many years ago were considered good reasons for not dredging. In other words, a dredge must be fitted to the ground and, in this manner the scope of dredging can be greatly widened. In buying a modern dredge, price is not the only thing to be considered; the dredge should be designed to meet the conditions under which it is to be used.

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* A discussion of jigs has been omitted because they have been used for 20 years or more and have been well described and discussed in the literature.

Mining Practices of the St. Joseph Lead Company in Southeast Missouri

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(St. Louis Meeting, October 1942, and New York Meeting, February 1943)

Southeast Missouri is the largest and oldest lead-producing district in the United States. For the year 1941, the statistical picture of pig-lead production, stated in short tons (partly estimated by the *Engineering and Mining Journal*) is as follows.

	Tons
World production	2,000,000
United States production	460,000
Southeast Missouri production	142,000

The consumption of primary lead in the United States in 1941 was approximately 900,000 tons, or almost double the country's production. The largest previous year's consumption in the United States was 718,000 tons, in 1926.

The largest quantities of lead are used for cable sheathing, storage batteries and white lead. During 1941 tetraethyl lead for auto and aviation gasoline consumed 50,000 tons of lead.

The entire lead production of Southeast Missouri is now mined by the St. Joseph Lead Co., whose dominant position as the largest producer in the United States of lead directly from ores is best shown by the fact that in 1941 it produced approximately 31 per cent of the lead production of the United States from its mines in southeast Missouri, where 22,000 tons of ore, averaging over 3.00 per cent lead, is mined per working day.

It is known that lead was mined at Mine La Motte, Madison County, Missouri, by Renault in 1723. Other white men and Indians probably did some mining prior to that time. Since 1865, the recorded production to Jan. 1, 1942, shows that the St. Joseph Lead Co. and the various other companies that have operated in St. Francois County mined lead ore, which, after milling and smelting losses produced 5,980,000 short tons of pig lead.

About 95 per cent of the present lead production of Southeast Missouri is mined in St. Francois County. The northern part of this area is 60 miles south of St. Louis. The remainder of the lead is obtained at Mine La Motte, about 25 miles southeast from the center of the producing area in St. Francois County.

Prospecting

In this district, most of the ore is discovered by surface and underground diamond drills and by prospect drilling underground with rock drills. Each 2-ft. length of the cuttings from these Jackhamer prospect holes is assayed for lead. These holes are marked and the records are entered on maps. It has been found that close drilling—100-ft. centers of holes, and sometimes less—is necessary in order to locate most of the smaller commercial ore bodies, many of which previously had been missed by drilling the areas with holes spaced 200-ft. centers or more.

To date, geophysical and other methods of prospecting, apart from diamond and

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churn drilling, have been of little value in this district in the location of ore bodies or the elimination of large areas of the Bonne Terre formation that can definitely be classed as not having ore.

The information obtained from these prospect holes is used for determining the size, grade and thickness of the various ore bodies that comprise the ore reserves of the mines.

These cores are estimated, or assayed, and the important parts showing the ore and character of the formation are stored for reference, since the data obtained are very important in any well-planned development program for the exploration of one or more ore bodies.

While there is always an element of chance in opening up new ore bodies, in this district the factor of realization value of the ore to that estimated is over 1.00, since most of the ore bodies actually exceed the area, thickness and grade of ore that is determined by the Engineering Department from the drill-hole records. These ore bodies, which mine out in excess of their estimated lead tonnage, more than counterbalance the occasional ore body that does not mine out as computed from the drill-hole records.

As an example of the comparatively large amount of prospect drilling being done, one representative group of mines that produces approximately half of the lead in the district averages per mine operating day about 2700 ft. of surface diamond drilling, 450 ft. of underground diamond drilling and 650 ft. of underground Jackhamer prospect drilling.

A recent outstanding improvement in surface prospecting is the abandonment of the steam-powered drill and the adoption of gasoline-powered diamond drills for all such work. These drills are trailer mounted and equipped with folding masts and an oil hydraulic feed. Improved swivel, double-tube core barrels have helped to increase the percentage of core recovery.

Ready set bortz bits are now used for both surface and underground diamond drilling. Carboloy bits are used only for drilling in sandstone. Solid bits average about 2500 ft. and core bits about 1200 ft. of drilling.

Twenty-five years ago, with the equipment in use at that time, an average of 30 ft. of drilling per 10-hr. shift was obtained. At present, an over-all average of 85 ft. per 8-hr. shift (including moves) is obtained. Recently 425 ft. of diamond drilling was cut by one drill in 24 hr. In another instance 230 ft. was drilled by a crew in 8 hr.; however, most of this was cut by solid bit drilling. Generally, 120 to 160 ft. of drilling with the solid bit or 100 ft. with the core bit can be run per 8-hr. shift in the shallower holes.

"A" rods and 10-ft. core barrels are used. Cores are 1⅛-in. diameter and 20-ft. rod pulls are conveniently made by the use of the new folding mast. About 12 gal. of gasoline is used per 8-hr. shift by each diamond drill on surface.

Coring is started, never less than 210 ft., and sometimes 275 ft. above the La Motte sandstone. Solid bit drilling is much cheaper than core drilling; however, it is very important that coring be started before the lead horizon is reached and that drilling be continued until the drill holes are bottomed in white sandstone, since recently, in some areas, comparatively large bodies of a good grade of ore have been found in the dark colored sandstone just above the white sandstone of the La Motte formation.

Development

At present, in this district, an average of about 0.6 ft. of drift, or its cost equivalent in other development advance, is being driven underground in order to provide sufficient ore in the stopes to produce each ton of lead in the concentrates. At one division, 26,630 ft. of drift and 920 ft. of

raise were driven during 1941—almost 30 times as much development by drifting as by raising.

More experienced study is required at present in maintaining a well-planned program of development than for any other phase of mining. The number of the long and the short developments should be so timed in a general way that new stopes are always ready to be placed in active operation whenever needed: for all stopes (some lasting 15 years or more) in time are mined out and must be replaced by other stopes having the necessary grade of ore for maintaining the cycle of profitable operations. Sufficient balanced development will prevent a mine from being placed in the position of having considerable ore reserves but so far distant from haulage routes that the tonnage of metal produced by the mine is often considerably curtailed, while long developments are being driven to their objectives.

In this district, comparatively large drifts, 9 ft. high by 10 ft. wide, are driven. This permits sufficient space for moving the St. Joe electric shovels over the tracks on the four-wheeled trucks especially designed for this work. These drifts allow the 4/0 trolley wire to be installed, a minimum height of 7 ft. above the rail and also give ample clearance for men between the passing locomotives and the power lines installed along the sides of the drift.

More drifts are driven by Conway electric shovels than by all the other shovels combined Nine air-powered shovels for use in drifts have been placed in operation in the last year. Portable scraper hoists have been in use for several years for cutting drifts.

The latest Conway loaders, type 40A, which have just been placed in operation, are equipped with a 40-hp. motor, steel tires on the cast-iron wheel centers, and have a conveyor clutch, which allows the conveyor belt to be loaded with nearly a mine car of rock while the loaded car is hauled away and an empty car brought back. It has a forward tramming speed of 100 ft. per min. and a reverse tramming speed of 130 ft. per minute.

Drifts are driven by contract, two men composing the crew on each shift. The normal cycle of operations consists of the two-man crew (one operating the shovel and the other the motor), first loading out the rock broken by the preceding crew. This fills generally from 15 to 20 of the standard 48-cu. ft. mine cars. After loading out, two vertical columns are set up and the round drilled out. Then the drilling equipment is torn down, loaded onto the shovel and moved back. The holes are charged and wired up and the round shot at the end of the same shift by the two men who loaded the rock. Many drifts average 5 ft. or better per round, although the average is about 4½ ft. when a round is made every shift.

From 26 to 32 holes are drilled per round, using about 15 lb. of 40 or 60 per cent ammonia gelatin powder per foot of advance. All firing is done electrically, using instants in the pyramidal shaped cut or "gun" and five different lengths of electric delays. The use of locomotive haulage has indirectly increased the drift footage, since fewer set-out tracks are required because they can be kept farther distant from the face of the drift, without loss of efficiency, than can be done where mules are used. It has also made possible the driving of drifts, to meet special conditions—some as steep as 5 per cent up or down, which could not be worked efficiently by mules.

While many drifts are advanced farther per 24 hr. than in the Lead Belt, the cost per foot of advance is low and an advance of over 2 ft. per man-shift is generally obtained.

The main factors, contributing to better results in drifting in the past 20 years, have been the elimination of hand loading by the use of mechanical shovels, larger cars,

FIG. 1.—ST. JOE SHOVEL.

FIG. 2.—CONWAY SHOVEL LOADING.

faster-cutting rock drills, smaller-gauge drill bits, locomotive instead of mule haulage, better ventilation due to more efficient direct-connected blowers and self-priming electric pumps for keeping down the water throughout the entire 24 hours.

System of Mining

All the lead of commercial value occurs in the ore as galena, which is disseminated, and also in thin layers or streaks, in the dolomitic limestone of the Bonne Terre formation. This formation is approximately 350 ft. thick and most of the ore occurs in the lower 100 ft. The ore is mined in open stopes, using the room-and-pillar system. No timbering or filling is required, since the roof, after being trimmed down, is, in most places, supported for many years by the pillars alone. The height of stopes is dependent on the grade of ore and, even though the galena in a stope may be entirely confined to a rich solid streak 2 in. thick, a minimum 7 ft. in height of stope is required for the present mechanized mining practice (Fig. 3). In some cases stopes are mined as high as 200 ft., although volumetrically the average height at present is about 14 ft. In a general way, pillars support the roof best when they are located or formed in a manner in which lines connecting their centers form approximately an equilateral triangle; however, this spotting of pillars is varied to conform with any change in direction of the trend of the ore, to more effectively support the roof near faults or channels and to leave as much as possible of the barren rock or lower-grade portion of the ore in the pillars. In most cases, the pillars are almost circular in cross section, varying in size from 10 to 50 ft. in diameter. The largest pillars are made in the highest stopes and where the worst roof conditions are encountered. In general, the pillars are spaced about 25 ft. apart, measured between their perimeters. In fact, pillars are spaced as far apart as the character of the roof will permit, since it reduces considerably the cost of breaking the ore and increases the percentage of recoverable ore. For the worst roof conditions, the spacing of the pillars is determined by the minimum clearances in which the St. Joe electric shovels are able to operate and a pillar spacing of 18 to 20 ft. is used.

Less than 1 per cent of the ore mined at this time is obtained by the partial slabbing or complete removal of the old pillars. In some places, groups of pillars have been removed after a series of concrete pillars have been poured between these old pillars. Only the richest pillars can be removed economically in this manner. Of total ore originally in mined ore bodies, 85 per cent has been mined and 15 per cent left in pillars. In almost all cases, since the pillars are mostly in the central part or rich core of the ore body, this pillar ore is richer than the average grade of ore mined in these stopes. These millions of tons of ore now in pillars constitute a large potential ore reserve, to be recovered when intensive underground prospecting has definitely shown that there is no additional ore in the backs or bottoms of the present stopes and that these stopes are not needed for haulageways to other ore bodies. At some future date, the solution of the problem of economically recovering these pillars, by safer and probably less costly methods than have so far been developed, should materially increase the percentage of pillars that can be profitably mined.

Breast stoping is a logical name for the method used in mining stopes 7 to 10 ft. high in the almost flat or slightly dipping ore bodies of this district. For mining stopes higher than this, a system of advancing the breast ahead of the bluff* is used and a 3 to 4-ft. bluff can be mined by the use of lifters alone. Where the bluff is too thick

* Bluff is a local term designating the part of the rock in place that is below the level of the breast and above the floor of the stope.

for this method, splitters* and lifters are drilled. In some high stopes, nearly vertical stope holes are used on one or more benches near the breast and the remainder of the bluff is drilled with splitters and virtually flat lifters for leveling the stope at the floor. More rock can be broken, at a lower cost per ton, by using lifters and splitters in bluffs than by using the almost vertical stope holes, with the resultant series of vertical steps or bluffs from the bottom to the top of the stope.

In some stopes, where a safe roof cannot be obtained without mining down a great deal of barren rock, in addition to the minimum height of stope required for closely mining the ore body, a system of supporting the roof by steel channels has been in satisfactory operation for some time. Steel channels (4 in. at 6¼ lb.) spaced as required, often 5-ft. centers, are held tightly against the lower side of the roof by 1-in. wedge bolts from 4 to 10 ft. long, driven into holes that are drilled at a 45° angle with the roof. The nuts on the lower, or threaded end, of these wedge bolts are tightly drawn up against an angle washer, which is placed against the web of the channel. These nuts are tightened with an impact wrench. The bottoms of the holes are always drilled deep enough to be in solid layers of rock and the bolts at the ends of the channels extend out over the pillars. This system prevents the layers of rock above the roof from separating from each other and falling down; aside from the safety viewpoint, this would lower the grade of ore in the stopes and increase the quantity of barren rock to be handled underground and later milled.

The cost of roof channeling varies a great deal, according to the condition of the roof. The mining cost is increased about 15¢ per ton of ore in stopes about 15 ft. high; however, the total cost per pound of producing lead is considerably reduced by this method. Better than average ore is obtained in the channeled stopes, since the richest ore often occurs where the rock is broken up and fractured considerably.

Because of the cost of roof channeling, it is used only in stopes where needed. At present, about 5 per cent of the ore mined in the district is produced from stopes in which the roof is supported by channeling.

The virgin ore bodies that are being mined today furnish the "reworked areas" for tomorrow. After a virgin ore body has been mined out apparently, it is prospected underground and often additional ore is found. In 1941, at Federal Division, 85.22 per cent of the ore mined was obtained from breasts averaging 13.65 ft. high. Backs and bottoms (reworked areas) furnished 14.77 per cent of the ore; their thickness averaged 9.75 ft.

Breaking of the Ore

All the ore is broken by one-man rock drills, mostly 45 lb., using a pneumatic feed either mounted on columns or laid on a lifter board. Side slabbing in breast drilling of the headings is still the standard method of breaking rock. An improved procedure is the drilling of multiple blind settings one behind the other and the shooting of from 6 to 24 or more holes in regular succession, one hole after the other, at one blasting time. This enables the heading to be more completely blocked out for the electric shovels and also allows a larger proportion of drilling time for the drillman; however, this type of drilling requires drillmen of superior intelligence, in order to drill the correct depth of holes and place a breakable burden on the points of the holes, since the failure of one set of holes to break would cause all the settings behind it to misbreak.

Virtually all rock is broken by contract; the contractor buying his powder, caps and fuse and being paid a price per ton for his rock dependent on the height of ground and whether it is back, bottom or breast. Base

* Splitters are holes drilled into the bluff above the lifters. They are pointed upward; the slope of the holes varies so as to give a breakable burden for each hole.

wages are guaranteed, regardless of the quantity of rock broken. The contract system is very popular with the men, since it is liberal enough so that average effort will earn a bonus and exceptional work will earn a large bonus. Contracting develops efficient rock breakers and good judges of the quantity of powder to be used in holes. The amount of rock broken per drill shift (provided the drillman cuts the lead ore cleanly and does not ruin his roof by drilling and shooting into it) is a much better yardstick of efficiency than the number of feet drilled per shift.

FIG. 3.—MINING IN HIGH STOPES.

The biggest improvement recently in rock drilling dates back about 10 years, when the gauge of the ⅞-in. hexagon drill-steel bits was reduced from 1⅝ to 1⅜ in. for starters with a 1/16 in. reduction in gauge for each 2-ft. change, making the last steel 1 1/16-in. gauge. This reduction in the size of the bits about doubled the speed of drilling. In order to get the powder to the bottom of the holes, ⅞-in. cartridges are now used instead of the 1⅛-in. formerly in use.

All drilling is done wet at present; bits average about 25 ft. of drilling before resharpening. Three-piece sets, using longer air feeds than the 30-in. air feed in the drifts, are being used more and more since the 3-ft. change is becoming so popular in the stopes; this allows the starters to have a 1¼-in. gauge with an attendant increased speed of drilling.

Detachable bits are automatically eliminated from consideration in this district, since the manufacturers have not been able to furnish bits of the small gauge used in the district.

It was found that the holes of smaller diameter broke virtually as well as those of larger diameter, and that it was not necessary to reduce the burden on the smaller holes. After changing over to the smaller holes, 60 per cent ammonia gelatin powder was substituted for the 40 per cent; after a short trial, the return to the 40 per cent powder was justified by the results obtained in this dolomitic limestone.

In the sandstone stopes of one mine 30 per cent ammonia gelatin is used, and gives better results in breaking than were previously obtained from the 40 per cent.

A low-grade 5 per cent equivalent powder, and also black powder, are used in shooting down the loose in trimming up the roofs of stopes.

One of the most popular and valuable aids recently to rock drilling has been brought about by the improved design of a rock drill featuring a steel lug, or boss, integral with the cylinder. The tapered end of the air-feed piston fits into this lug and is held in place by a nut on the threaded end of the piston. This boss has eliminated the rather heavy and loosely fitting clamps around the rock drill that supported it by being fastened to the end of the air-feed piston. The hose connection from the rear of the machine to the air feed has also been omitted, since a small, manually operated valve, conveniently located on the outside of this boss, gives the drillman close control of the amount of air required by the air feed to suit all the variable conditions of drilling. The use of 2-in. standard pipe columns and lightweight welded column clamps has materially helped the drillman.

For increased safety, an electric light in an enclosed cylindrical container is installed near the working face as close as blasting will permit, and if a miner's cap light fails it gives a ray of light to guide him by the shortest route to a safe place. Wire fuse igniters are used almost entirely. They afford a faster, more certain and safer method of spitting the fuses.

The contract drillmen in the stopes do the mining down of the loose rock in their headings, and average 55 tons of rock per drillman shift. Breaking in different stopes varies from as little as 30 tons to as much as 120 tons per drillman shift (exclusive of pillar removal). This tonnage depends principally upon the height of ground, character of the roof, spacing of the pillars and in a great degree to the rock-breaking ability of the individual drillman.

More men are required as drillmen than for any other occupation. At Federal Division, 33⅓ per cent of all the payroll shifts underground are drillmen in the stopes and drifts; just half of the drift contractors being counted as drillmen, since virtually half of their time is spent in drilling.

The breaking of sufficient rock to supply the mills with a uniform tonnage of the desired grade of ore every day requires much more time and supervision than the

loading, hauling and hoisting of the ore, since there are more variable and changing conditions in the stopes.

At one group of mines, an average of about 0.55 lb. of 40 per cent ammonia gelatin is used per ton of rock broken in the stopes.

Loading

Virtually all the ore in the Lead Belt is loaded mechanically. The change from manual to mechanical loading led to a large saving in operating costs, which has partly tended to counterbalance increased wages, higher taxes and a lower grade of ore. Hand loaders formerly averaged about 20 tons per shift while St. Joe electric shovels, exclusive of time required for moving, average 175 tons per shift under present conditions.

The superstructure or deck of the St. Joe shovel is mounted on a continuous-tread truck (Fig. 4). This shovel has three motors: the crowd and the hoist motors are 20 hp. each and the swing motor is 12 hp. These motors furnish the power for the various motions of crowding into or retracting from the ore, hoisting, swinging and traveling of the shovel. This shovel can load in a space 7½ ft. high and swing between pillars as close as 18 ft. to one another. These are two of its outstanding features. In addition to being a very rugged and dependable shovel, it has low maintenance and operating costs. It can load at an average rate of about one ton per minute; however, only about half of the total elapsed loading time can actually be used because time is required to switch the cars back and forth from the headings to the loop.

Fig. 4.—St. Joe shovel loading.

Another feature, which especially makes the St. Joe shovel outstanding over all other types for use in this district, is the ease and speed with which it can be loaded by its own power onto a low St. Joe shovel truck and hauled over the tracks to the next stope, then unloaded and run into the heading ready for loading. This complete operation generally requires about 2 hr., even though some shovel moves are 2 miles or more distant.

Bonus is paid to both men of the two-man crew, the shovel operator and the driver, for all tonnage above a fixed amount, or score, and the necessary time allowed for moves to other stopes.

At one group of mines having a central hoisting shaft, all the loading and hauling has been mechanized. In the stopes, 32 St. Joe electric shovels are used and in the developments, 32 drift shovels, air and electric.

Two-drum and three-drum electric scraper hoists are in use; ⅝-in. hoist rope and 54-in. scrapers are standard.

Two different types of remote control have been developed for the scraper hoists. This remote control allows the operator to stand where he can easily see both the loading and discharging of the scraper and work much more efficiently and safely than standing behind or to the side of the scraper hoist while operating it.

The old-style, manually operated, arc gate, used for loading ore from the underground chutes into mine cars at the bottoms of raises, has been superseded by an efficient and easily operated wooden finger gate, which has a small steel arc gate separately controlled at the lip of the chute. This helps to regulate the flow of ore and reduce the quantity of spillage while cars are being loaded.

Haulage

During the past 25 years, there has been a continuous improvement in the efficiency of underground haulage. Twenty-five years ago, there were numerous shafts in the district, each equipped with its own surface crushing plant. The crushed ore was hauled from these separate shafts in railroad cars to the nearest mill.

Our experience has shown that it is cheaper to haul underground to a centrally located shaft at the crushing plant of a large mill than to maintain a number of small surface crushing plants and haul this crushed ore by surface railroad to the nearest mill.

This trend of having fewer but larger mills, centrally located with respect to the mines, has resulted in longer average hauls from the various mines but has permitted a consolidation of shops, milling, etc., that has more than compensated the cost of an increased length of haul.

In the early days of mining, mules, either singly, or as many as four in line, were used to pull a trip of the small one-ton mine cars to the shafts; the distances, at that time, were comparatively short. Later on, high-pressure compressed-air locomotives were installed on the main lines and reduced the cost of hauling. This type of haulage in turn was completely replaced by storage-battery locomotives, and while these were an improvement over the compressed-air locomotives, they were not well adapted for the heavy trains and long distances; so they in turn were completely superseded by the trolley locomotives that are in use today. In most of the mines, the mules have been entirely replaced in the headings and drifts by reel locomotives and the haulage has been completely mechanized, which has resulted in further economies (Fig. 5).

The average length of haul for all the mines in the Lead Belt is about 2 miles. At present, haulage costs more than any other operation underground, but the additional cost of haulage per ton-mile due to increased lengths of haul increases much less proportionately than the additional increase in the average length of haul.

In 1941, at one group of mines, 3,000,000 tons of ore were hauled to a central shaft, an average distance of 2.76 miles and 0.90 kw-hr. was required per ton-mile of ore hauled.

FIG. 5.—HAULAGE TRACKS IN HIGH STOPES.

Tracks of 24-in. gauge and $23\frac{1}{2}$-in. wheel gauge are used at all the mines except one, which has 30-in. gauge tracks. The benefits of changing from 24-in. to a wider gauge, at this time, would not warrant changing the gauge of hundreds of miles of underground tracks, nearly 200 locomotives, more than 4000 mine cars, mechanical drift shovels, rotary dump cars and other equipment.

Our primary main-line tracks, representing our best underground track construction, are built with creosoted ties and tie plates. Rail braces are used on the tie plates on the outside rails of the curves.

The track beds are kept well drained by water ditches alongside the track, generally on one side. After new track has been hauled over long enough to be settled, it is ballasted and tamped to grade, using chat or fine rock. This is very important, since tracks that are laid directly on a solid rock bottom do not stand up well. The rail joints are then electrically welded to lessen track, car and locomotive maintenance by having smooth, rigid rail connections, and to provide good electrical conductivity for the current returning to the motor-generator sets.

Because of the uniform temperature underground, long satisfactory service is obtained from the welded rail joints; in fact, the brazing of copper bonds at rail joints has been virtually discontinued.

No. 5 welded frogs, using 60 or 70-lb. rails, are made underground and are standard for main lines; 30-lb. rails and No. 3½ frogs are used in the headings.

The standard mine cars have a capacity of 48 cu. ft. level full, weigh 3600 lb. and hold an average load of 5400 lb. of ore. Other specifications are 23½-in. wheel gauge, 28-in. wheel base, 16-in. cast steel wheels, top of car body 41 in. above rail, swivel couplings, outboard half-section bronze axle bearings with Satco lining, which is a lead alloy giving excellent service.

Newly designed cars of 14 tons capacity are being tested. They will increase the net pay load hauled from 60 per cent for the 2.7-ton cars to 75 per cent for the large cars. Eleven of these cars have been in service more than a year and are giving lower costs per ton-mile of haul.

Because of the large amount of new tracking continually required in the headings, the average cost of constructing and maintaining the heading tracks at one group of mines, having an average haul of 2¾ miles, is double that of the main-line tracks. The total cost of maintaining all the main-line locomotives is just half the total maintenance cost of all the reel locomotives.

Rubber-tired, trackless haulage, when available, warrants a trial in the headings, both to decrease haulage costs and to decrease the quantity of ore left on the floor of the stopes.

Standard size of locomotives for the headings is 8 tons. The reels hold about 600 ft. of No. 2 single-conductor rubber-covered cable. These locomotives are large enough to haul a St. Joe electric shovel over the tracks when it is loaded on its shovel truck, and to handle two or more loaded cars on the steepest haulage grades, some being as steep as 5 per cent up or down.

For main-line work, 15-ton electric trolley locomotives are standard. These have two 90-hp., 250-volt, d. c. motors, with ball bearings at both ends. They are equipped with two trolley poles having gliders. Thirty-two-volt headlights, with dimmer switch, are used and a controller with series and parallel. Blowers of ⅓ hp. are used for the motors. Because of the 23½-in. wheel gauge, reach gears are used between the motor pinions and the axle gears. Hydraulic, dynamic and hand brakes are used. Steel tires, 33-in. dia., are used with steel centers; wheel base is 66 inches.

On one main-line haul, 2¾ miles long, with a grade in places of 1.80 per cent against the loads, a 25-ton locomotive is used. The controller has 10 points and operates in parallel only, no series being provided. The contactors are of the semi-magnetic type. The 150-hp. motors are cooled by separate ⅓-hp. blowers. In addition to hand brakes, air brakes are used. They are very satisfactory and well liked by the motormen.

Standard copper trolley wire is 4/0. Bare copper feeder wire 300,000 cm., or 1,000,000 cm., as the case may be, is supported by a

clamp directly above the bulldog clamp that supports the trolley wire. This is a decided improvement in efficiency and cost over the previous method of having a rubber-covered feeder wire supported on the side walls and connected at intervals by a knife switch and jumper wires to an ear on the trolley wire.

FIG. 6.—SAMPLING ROOM AND ROTARY DUMP.

Since 1937, gliders have replaced trolley wheels on all locomotives weighing more than 8 tons.

The trolley wire is quickly conditioned whenever needed by applying a graphited lubricant by means of a special trolley-wire lubricator installed on a separate trolley pole mounted on a locomotive. The amount of lubricant applied is regulated by an adjustable wiper.

In hauling over a single track, locomotives are protected from running into one another by a system of electrically operated red or green signal lights at the ends of each block. Where the traffic is dense, the extra cost of automatic block signals is justified, considering that in most cases more safety is obtained. Hand-operated electric signals, while probably not possessing as high a degree of safety, are more cheaply installed and give fairly satisfactory service.

Recently, an additional safety feature has been added to the hand-operated signal blocks, in the form of a separate transformer and a separate cable for each signal block. In this way, any electrical trouble in one block is not transmitted to the other signal blocks, and a grounded wire in one block cannot change the color of light burning in another signal block.

Six-conductor No. 14 rubber-covered, coded, copper-wire cable is used for the hand-operated signal blocks.

Motor-generator sets, furnishing direct current at 275 volts, are located at various places, so as to equalize the voltage as closely as practicable over the haulage system. These sets vary in size from 100 to 300 kilowatts.

Three-car, electrically operated rotary dumps are standard practice for dumping the ore or poor rock into the skip pocket at the various hoisting shafts. One of these underground ore pockets, having two rotary dumps, has a capacity of over 1800 live tons of ore.

At the largest shaft, all loaded and empty cars are automatically weighed on a Streeter-Amet weighing machine and the cars are automatically sampled while being dumped (Fig. 6). In this manner, each of the seven mines hauling to this central shaft is credited with its own tonnage and grade of ore produced every shift. Pounds of lead per machineman, per shovel operator and per mineman, and tons of pig lead equivalent per day for each mine, are stressed far more than tons of ore per man in calculating costs and efficiency and giving the mine captains their daily operating report.

Hoisting

A short description of the hoisting at No. 17 Federal shaft will be given, since it is the largest and most modern hoisting installation in the district. The new hoist started operation on Oct. 6, 1938. The cost of the hoist and motor, f.o.b. factory, no installation costs or control equipment included, totaled $32,500. The ore is loaded from an 1800-ton live-capacity pocket into skips, averaging 8¼ tons; these operate in balance in a vertical shaft.

The flow of ore into the skips is controlled by two hydraulically operated gates at each loading chute. Each front gate is of the undercut vertical type operated by a 10-in. dia. cylinder directly above the gate. About 6 ft. horizontally behind this gate is a counterbalanced arc gate operated by a 6-in. cylinder. Water at 225 lb. pressure is used in the cylinders to operate these gates.

The skip-loading station is on the opposite side of the shaft from the chutes. Here the skip loader, conveniently seated, can see and regulate the quantity of ore loaded into the skips by operating four Critchlow, four-way valves that control the water for the hydraulic cylinders—a 3-in. valve for each 10-in. cylinder and a 2-in. for each 6-in. cylinder.

All the ore is hoisted from one level. The vertical distance from the loading to the dumping positions of the skip if 545 ft. The maximum rope speed is 1585 ft. per min. The total time required for hoisting a loaded skip is 28.8 sec., consisting of 6 sec. each for acceleration and retardation and 16.8 sec. running time. With average loading ore, about 4 sec. is required to load the skip, making a complete hoisting cycle of about 33 sec. While this hoist will meet the manufacturer's guarantee of 109 trips per hour, the best hoisting and lowest maintenance costs are obtained by not exceeding a rate of 100 skips per hour, which is equivalent to about 825 tons per net hour of hoisting, because at present there is considerably more hoisting than milling capacity.

The center of the 8-ft. dia. cast-steel sheaves on the steel headframe is 101½ ft. above the collar of the shaft. Bronze-bushed 8½-in. dia. ball and socket bearings support the sheave shafts.

A 1000-hp., 450-r.p.m., 440-volt induction motor with herringbone pinion forged integral with the 8-in. dia. motor shaft drives the cylindroconical drum a maximum of 46 r.p.m. The general design of the drum is 6½-ft. cylindrical end portions joined to an 11-ft. cylindrical center portion by steep cones.

The oil-braking system, operating the post brake, is provided with an automatically operated accumulator and two separate oil pumps. A Lilly type D governor is used.

The location of the hoist gives a maximum fleet angle for the drum of 1°04′ and 0°28′ for the sheaves.

Hoist ropes 1⅝ in. thick are used 6 × 19 × 900 ft. long, modified Seale construction, regular lay. The best rope to

date hoisted 2,168,000 tons of ore and the last three ropes averaged 1,944,000 tons each at a rope cost of 17.9¢ per 1000 tons of ore hoisted.

Power

The Union Electric Light and Power Co. furnishes all the electric power used by the St. Joseph Lead Company's mines in St. Francois County. Power is transmitted at 132,000 volts from Bagnell Dam and Cahokia, either or both as required, to a central distributing station at Rivermines, Mo., where the Company has a modern power plant using pulverized coal for producing steam for the turbogenerators. This power plant operates only when required; that is, in emergencies and during the low-water periods of the year. From this central plant, power is transmitted at 6600 volts to the various mines and other places where it is used. It is transformed down to 440 volts before being taken underground.

The peak-demand power load for the St. Joseph Lead Co. in St. Francois County averages about 25,500 kw. and the total load is about 540,000 kw-hr. per day.

During 1941, an average of 25.27 kw-hr. was required per ton of ore treated. This included mining, milling, and other operations. Operating and development mining required an average of 6.04 kw-hr. per ton and pumping an additional 4.93 kw-hr. per ton, making a total power consumption of 10.97 kw-hr. per ton of ore mined.

The mining operations (exclusive of drainage) accounting for the largest amounts of power in 1941 were.

	Kw-hr. per Ton of Ore
1. Haulage	2.0
2. Compressors for drilling	1.8
3. Loading	0.6
4. Hoisting	0.6

Miscellaneous

Portable compressors, using both alternating and direct current, mounted on trucks, are used underground.

About 5000 hp. is required to operate the surface compressors during the peak hours of operation in the Lead Belt. Compressors are set to cut out at 105 lb. and cut back in at 100 lb. pressure.

The use of electric cap lamps is being extended, as they are far superior to carbide lighting.

Working places are sampled regularly by the dust-control department; however, as wet drilling is used almost exclusively, the dust counts are away below the safe permissible standards.

Drinking water is sampled underground and all places are marked where the water is unfit for drinking.

Picture show places are maintained by the Safety Department, both on the surface and at many places underground.

With the large number of shafts and churn-drill holes in the mines, artificial ventilation, assisted by blowers on the surface, provides an abundance of fresh air underground. Remote places not ventilated by a churn-drill hole, are ventilated by small direct-connected blowers through canvas or galvanized pipe.

Almost ideal working conditions are present underground. The average temperature, short distances from the shafts, is about +61°F. and the relative humidity is about 90 per cent.

These conditions attract and hold a very fine type of steady workmen. The labor turnover is very small; the majority of the men continue at their work until they are 65, when they become eligible for pensions.

Mine Drainage, Southeast Missouri Lead District

By W. W. Weigel,* Member A.I.M.E.

(St. Louis Meeting, October 1942, and New York Meeting, February 1943)

The mines of the St. Joseph Lead Co. in St. Francois County, Missouri, form a roughly triangular area of about 45 square miles. Locally this is known as the Lead Belt. The four operating mines in the area at present are the Federal, Leadwood, Desloge, and Bonne Terre, in decreasing order of mine area and daily production. Mining began in 1864 and has been carried on in the district continuously since that time.

As a group, the Lead Belt mines now handle 20,000 gal. per min. of water, average head about 465 ft., against a normal daily ore tonnage of 21,200 tons. For a six-day operating week this gives a ratio of 6½ tons of water per ton of ore. Total average pumping in 1931 was about 15,000 gal. per min. Since that time a large area of virgin territory has been opened, much of which has had temporary flush flows. About 75 per cent of the water enters in certain restricted very wet areas. Production figures are given in Table 1. The figures by divisions for 1931 and 1941 are not exactly comparable, as there were several transfers of pumping loads from one division to another during that period of time.

Table 1.—*Production Statistics, Lead Belt*

Division	Water, Gal. per Min.		Average Pump Head	Daily Tonnage of Ore	Water-ore Ratio
	1931	1941			
Federal	8,000	11,000	475	11,000	7.0
Leadwood	4,790	5,900	450	4,500	9.2
Desloge	510	1,200	320	3,500	2.4
Bonne Terre	1,675	1,530	475	2,200	4.9
Totals and average	14,975	19,630	465	21,200	6.5

Geology

The basement formation in the Lead Belt area consists of the pre-Cambrian rhyolite porphyry and granite. Overlying this is the La Motte sandstone, of Middle Cambrian age, which offers a good medium for circulation of water. It varies in thickness up to 400 ft. Above this is the Bonne Terre dolomite, approximately 375 ft. thick, which in turn is overlain by the 150-ft. Davis shale. The formations above this, present in the Lead Belt in small areas only, are of minor importance. The ore bodies occur as large, horizontal tabular masses or as long, narrow runs; they are found throughout the Bonne Terre but largely in the lower 150 ft. The rather indefinite contact with the La Motte is a favorite ore horizon. Mining is entirely by open stope and pillar breast systems.

To the south, west, and north of the district, sandstone outcrops are few and limited. Heavy faulting bounds the district on those sides and probably there is little if any artesian circulation from those directions. To the east, the La Motte rises to an outcrop area of about 40 square

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* St. Joseph Lead Co., Leadwood, Missouri.

miles, 8 miles east of the district. As this outcrop area is well drained by surface streams of normal size, and the La Motte is either thin or missing under a considerable part of the ore area, the amount of artesian flow from this direction is probably small.

Direct seepage from the rainfall over the mine zone is slight. About half of the Lead Belt is overlain by the Davis shale, a very impervious bed, and much of the rest is covered by residual red clay from 10 to 20 ft. thick. Mine workings near the surface at Bonne Terre and Desloge are commonly very dry and show no oozing or dripping even after long wet seasons.

The Bonne Terre dolomite in itself is a poor water medium. It is broken up to a high degree, however, by systems of joints, slips, fracture planes, and other openings, which not only circulate water but act as a reservoir. Most of these openings have been caused by differential settling of the La Motte sandstone around the high-porphyry points, leading to faulting on a minute scale, and run somewhat at random.

Several well-defined channel zones (Fig. 1) two or three miles long have been located. Each of these has a persistent strike, not all the same, and may be from 100 to 500 ft. wide. The strike of the major openings, some up to 4 ft. wide, is usually about 50° from that of the zone itself. The channels running across the zone may be from a few inches to 100 ft. apart along the strike of the zone and are cross connected by a network of small, irregular openings. Only slight faulting is evident.

FIG. 1.—CHANNEL ZONES AND FAULTING, NORTH END, LEADWOOD.
SCALE: 1 INCH = 2500 FEET.

The top 15 to 30 ft. of the Bonne is especially prone to shattering and formation of cavities. At 75 to 100 ft. from the top there is a layer characterized by many horizontal openings, which probably is the main circulating layer of the district. Between these two beds there are vertical channels. The lower part of the Bonne Terre has only vertical channels, many of

which connect through to the sand below. A generalized sketch of channels and mine workings at the Leadwood mine is shown in Fig. 2.

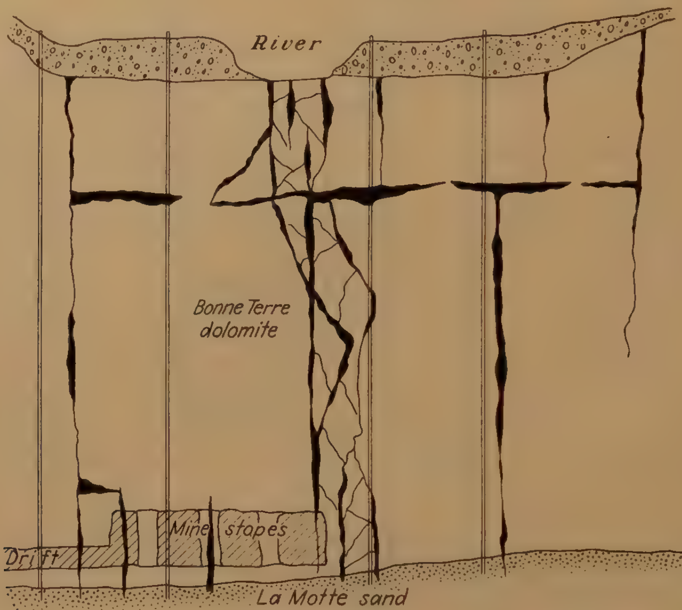

FIG. 2.—GENERALIZED SKETCH OF CHANNELS AND MINE WORKINGS, LEADWOOD MINE. NOT TO SCALE.

Big River, the main drainage of the area, takes a winding course of 25 to 30 miles across the district. Where the channel zones cut across creeks and rivers, or the two upper layers of the Bonne Terre mentioned form outcrops under drainage lines, there is a chance for considerable addition to the ground-water system. A drift being run from Leadwood in 1927 cut through about 200 ft. of one of the channel zones and made 5000 gal. per min. of water. This was traced to the outcrop of the zone, about 300 ft. long beside the river, about ¼ mile away from the drift. The river bed, of rock, was a network of large open joints, up to 12 in. wide. Fluorescein poured into one of these showed up in the drift in a few minutes. The extreme drought of 1936 gave a good opportunity to observe this type of opening in the bed of Big River. As the river gradually dried up, one opening after another would take the entire flow to that point. At one place, 2 miles west of any workings, about 1000 gal. per min. was disappearing. Many of the small creeks leaving the Davis shale outcrop lose much or all of their water flow when running over the outcrops of the two porous horizontal layers near the top of the Bonne Terre. As their flow varies with rainfall, this inflow is not steady. Up to 500 gal. per min. is lost in some places.

After the water gets into the underground circulation, either through the channel zones or through the outcrop of the two horizontal layers, it travels to vertical channels or to one of the 50,000-odd diamond-drill holes in the district. These let it down either into workings or into the sand, from which it will rise to the workings by other channels and drill holes.

Pumping Equipment

Owing to the long period of time over which the mines were developed, and the fact that they all consist of consolidations

of several mines (many of them opened by different mining companies), there is a great deal of difference in pumps, pumping-station layouts, and drainage systems. General pumping equipment at the end of 1942 was as listed in Table 2.

TABLE 2.—*General Pumping Equipment at End of* 1942

Division	Shaft Pumps				Relay Pumps		
	Stations	Pumps	Horsepower	Total Capacity, Gal. per Min.	Stations	Pumps	Horsepower
Federal	4	12	3,800	24,600	10	20	1,175
Leadwood	7	20	3,350	18,680	13	21	312
Desloge	2	7	965	5,000	8	10	130
Bonne Terre	2	6	875	3,600	10	13	115
Total	15	45	8,990	51,880	41	64	1,732

During the past few years, there has been considerable relocation and consolidation of main pumping units to improve general pumping efficiency. By the use of churn-drill holes for pump discharges, located to minimize the static head, considerable power has been saved. The Pimm mine, formerly worked by the National Lead Co., was consolidated with the Federal mine. A much lower surface elevation on the near-by Federal land enabled the sump to be relocated and the head cut from 714 to 486 ft. At Leadwood recently, the opening of a new station almost a mile from an old shaft station, with some small development, eliminated four relay pumps and reclaimed 3000 ft. of pipe lines.

The 45 shaft pumps, all centrifugal, consist of 20 different types, makes and sizes. The older pumps, many of them inherited from other companies, are maintained as emergency spares and are run only enough to keep them in shape. Most of these are controlled by hand. The unit commonly in active service is a 200-hp., 1000-gal. per min. rated capacity, 1760-r.p.m., two-stage Allis-Chalmers Hydracone, with the impeller sized to suit the particular head and with full automatic control.

The most elaborate pumping station is that at No. 2 Doe Run shaft of the Federal Division, which handles about three fourths of the water of the Federal mine. The pumps are set on a concrete floor covering a drift from the 21,000,000-gal. sump, and the suction pipes are run through stuffing boxes, as the sump overflow is 11 ft. above the pumps. The main units are three Allis Chalmers 12 by 10-in. two-stage pumps rated at 4800 gal. per min. at 435 ft. head and the 1200-r.p.m. motors are E. M. Co. 650-hp. 440-volts, two synchronous and one induction. A fourth pump, 1000 gal. per min., completes the station. These are both full automatic and hand-controlled. With its large sump storage, this station is used as a power control for the whole district and power peaks may be averted by shutting it down temporarily.

BULKHEADS

Until recent years, no systematic effort has been made to prevent ingress of the water. A few channels were caulked and wedged, at best only a temporary measure. Some of the diamond-drill holes cut by the mine workings run a strong stream of water. If these are merely plugged, the water pressure tends to loosen the back around the hole. Grouting the hole either from the top or bottom will stop the flow usually.

Bulkheading of abandoned stopes has been little used. In the past mining was largely either in or near the large central ore zones, where almost all the workings

are connected by large open stopes, impossible to block off. Mining is now tending to more isolated areas and as these are worked out and are not needed as a route to reach other ore bodies, the connecting drifts may be bulkheaded and the water shut off.

In 1927, two emergency bulkheads were built at Leadwood to stop a flow of 5000 gal. per min. from a single drift. This was the inflow through a channel zone from the river, mentioned before. These bulkheads were 6 ft. thick, heavily reinforced, and keyed 6 ft. into all the walls of the 8 by 10-ft. drift. Total size was 22 by 20 by 6 ft. and they were designed for a total head of 340 ft. Doors 4½ by 6 ft. in size were swung from above the doors on the pressure side and fitted against rubber gaskets.

In 1936, an area about a mile square north of the Baker shaft, worked by the Federal Division, was to be abandoned for several years. This area was making about 600 gal. per min. and was shut off by three widely separated bulkheads.

An isolated area of about 160 acres was mined out by Leadwood by 1939. This had only one connecting drift to the rest of the mine. A bulkhead shut off a flow of 400 gal. per min. These last few bulkheads are simply concrete plugs, filling the drifts for twice the distance calculated to resist the shear along the walls at 35 lb. per sq. in. The last one mentioned was 9½ by 10 by 17 ft. No keys for the concrete were cut, as the rock walls of the drifts have enough irregularities. Several small openings were shot up into the back and 1-in. or 2-in. pipes led from these to the outside of the forms. The drift water was piped through the forms. The concrete was placed with a concrete blow gun and, by using a stiff mix, the space was filled virtually to the back. After a few weeks, the edges of the concrete were caulked and cement grout was pumped through the small pipes and a good seal obtained. After about two weeks more, the water pipe through the bulkhead was valved off. Five weeks was spent on this last bulkhead. Cost was about one third of the door type. Leakage is nil with a head of 100-lb. on the bulkhead.

Sliming

Because of the size and number of the broken outcrops, laying concrete slabs in the river bed was out of the question. The river has a flood area over gravel beds about 800 ft. wide in places. In 1930, cement grouting of the individual channels was attempted but 400 sacks pumped into one channel made no impression and many others were to be pumped. The Leadwood mill, 8000 ft. away from this point, was producing about 1500 tons a day of flotation tailing (spoken of as slime) which could be used for grouting. A 4-in. pipe line (Fig. 3) was laid to carry this tailing from the mill to the area that had lost the 5000 gal. per min. and had been bulkheaded off.

Prospecting in the southeast Missouri area is done with the diamond core drill. Holes are spaced from 50 to 400 ft. apart. A 3-in. pipe is set as a sleeve to the top of the rock, to keep soil and gravel out of the hole. If no trouble is encountered, the hole is then drilled to the top of the La Motte sandstone. The hole diameter is about 2⁵⁄₁₆ in. If necessary, 2½-in. casing pipe is set inside the 3-in. and may run some distance into the rock, with a more or less tight fit.

The slime grouting of the channel areas was done through the holes already drilled in the neighborhood and through several pitch holes drilled across the strike of the channels. Most of these holes took large amounts of slime for several weeks, and, as the water flow down many of the channels apparently stopped, the results were considered good. For economic reasons, however, no more work was done from 1930 to 1936. Since that time, the work has been continuous except during the winter seasons.

The slime as available at the mill runs 25 per cent solids, of which 16 to 17 per cent is on 100 mesh. The coarser material settled badly in the long pipe lines and caused many stoppages. By a partial classification, producing a feed of 20 per cent solids and only 8 per cent on 100 mesh, and by maintaining a velocity of at least 150 ft. per min., the pipe-line stoppages have been largely avoided. Table 3 shows the composition used at present.

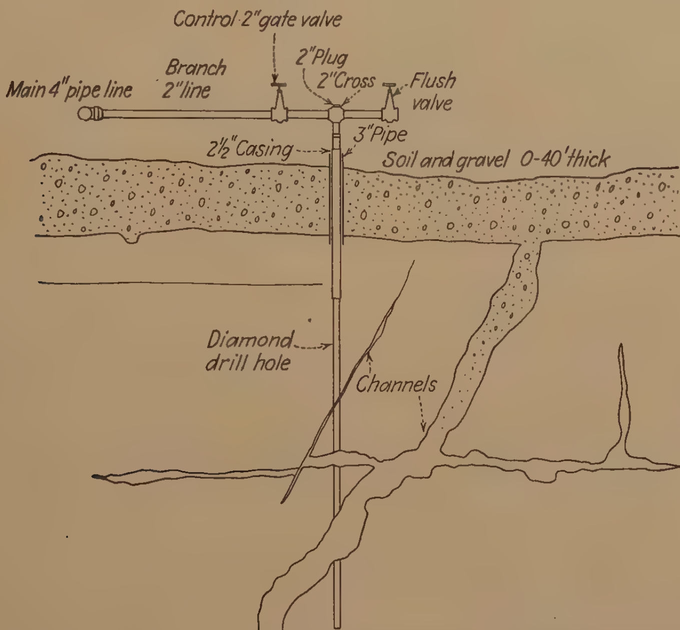

FIG. 3.—GENERALIZED SKETCH OF SLIME PIPE TO DIAMOND-DRILL HOLES, LEADWOOD MINE. NOT TO SCALE.

TABLE 3.—*Composition of Grouting*
PER CENT

Mesh	River Slime	Mill Flotation Tailing
+ 48	0.3	1.0
+ 65	1.2	4.2
+100	6.6	11.7
+150	11.2	14.4
+200	15.1	16.6
−200	65.6	52.1

The material chemically runs about 85 per cent dolomite. The remainder consists largely of sand, a little FeS_2 and small amounts of various silicates, of which glauconite and various kaolin minerals are the most common. No actual colloids are present. The material will settle clear in a comparatively short time.

The diamond-drill records indicate the channel zones and porous areas fairly closely. This is helped by the location of sink holes, study of the river and creek beds in time of low water, location of channels in the mine and other factors. The area to be filled may be selected either as the part near the river, where the water is introduced, or around the workings, or both. Almost complete filling is essential. Extra drill holes usually are necessary. In the sliming area, 2-in. lines branching from the 4-in. line are run to individual drill holes where a threaded connection is made through a control valve and a cross tee. From 30 to 50 lb. pressure is available at the top of hole. More is not necessary, and after the slime once ch_kes in the hole

and begins to pack, no pressure will force more in.

Many of the holes take slime so freely that they develop a strong pull from the slime falling down the hole, and are called "suction holes." Others take slime only under pressure. Some of the suction holes have taken as much as 10 tons of dry slime per hour and have run for several months, the amount gradually decreasing. The record hole took about 5000 tons of solids over four months time. Many choke off in a few hours.

Breakouts of the slime under pressure to the surface may occur around the sleeve or up through connecting channels to a distance of several hundred yards. Many break out through other holes or up in the bed of near-by streams. After discovery of the breaking-out point, the control valve is regulated to minimize the surface leakage.

The number of holes connected at one time varies from 4 up to 30. As some holes fill up, others are added, filling out and working across the area. The holes in any one group may choke off at about the same time but any sudden stoppage of a strong suction hole is due usually to caving and blocking of the hole. The diamond drill is brought back to such holes and they are cleaned out for a second treatment.

Up to Jan. 1, 1943, grouting had been run into 763 holes over an area of 240 acres. Approximately 360,000 tons of solids has been pumped. As the slime may travel long distances, having been found underground 2000 ft. away from any slimed holes, the actual area affected is much greater. The settled slime in the openings acquires the consistency of very stiff putty. Diamond drills have cut out cores of this material 6 in. long.

The sliming is not intended to fill every opening completely; 90 to 95 per cent would be a good average. Any water coming through the remaining openings may then be shut off by a small amount of grouting. Channels cut by mine workings before treatment would show 125 to 150 lb. pressure. No channels have been cut in a thoroughly treated area that showed more than 30 lb., and usually they showed less. This water seems to come from the sand below the workings. There has been no evidence of any surface water in quantity leaking through in any slimed area after treatment.

A small part of the mine made about 200 gal. per min., all as a general oozing and dripping. About 30 holes in the neighborhood were treated over about three weeks, which cut the water to about 15 gal. per min. The strong channel zone that put 5000 gal. per min. of water into the mine in 1927 has been the most intensively slimed. Recent drifts put through this zone have cut only small flows easily grouted off. Test holes are kept ahead of the faces to give warning of channels and allow grouting. Seldom is more than 100 sacks of cement required, many channels using only 10 or 20, as compared with as much as 1200 sacks per channel before sliming. The use of Aquagel—processed bentonite—as a 4 per cent mixture in the cement grout eases the pumping and helps the grout to seal better. Such a mixture does not shrink on setting. Unless prepared mixtures are bought, the Aquagel requires mechanical mixing. At Leadwood, the cement, Aquagel and water are mixed to a smooth slurry by using a sump pump set in a small tank as a circulator. Aquagel has been used also to pour down drill holes from which unsettled fresh slime was breaking into the mine through small cracks. Its swelling action soon stopped these leaks. The channels when cut are filled with masses of red clay, settled slime, and a small amount of grout, which has acted as the final plug.

Indirect evidence shows that the blocking of surface leaks may cut down inflow of mine water in distant parts of the mine to some extent, by lowering the pressure in the sand. A new surface settling pond began to lose about 1500 gal. per min. of water into the

ground, as its area extended over a channel zone. An increase of water of only 100 gal. per min. was noted in the nearest workings. A small amount of sliming along the edge of the pond for two weeks sealed the flow of the pond and stopped the leakage. The inflow of 100 gal. per min. stopped in the same week.

Direct charges against the sliming are the power and maintenance of a 75-hp. pump, pipe and fittings, and three operators (one per shift). The extra diamond drilling necessary has been charged usually to prospecting. As handling of the slime to a settling pond would be necessary anyway, pumping costs are not a direct loss.

It is probable that this method could be applied in other districts where the inflow of water is through fractured material. For shallow mines, the same general system would apply, using either diamond-drill holes or churn-drill holes. In one place at Leadwood, slime was piped down a churn-drill hole and 15 underground diamond-drill holes were treated. Choking these required three weeks time. Deep mines could use a similar system.

Developing, Milling, and Smelting the Ores of the Tri-State (Missouri-Kansas-Oklahoma) District

By George M. Fowler,* Member A.I.M.E., R. E. Illidge† and B. M. O'Harra,‡ Member A.I.M.E.

(St. Louis Meeting, October 1942 and New York Meeting, February 1943)

Part I. Developing

By George M. Fowler

Introduction

The Tri-State district comprises an area of about 2000 square miles in southwestern Missouri, southeastern Kansas, and northeastern Oklahoma. The part that is in and contiguous to Missouri is still sometimes designated as the Joplin district (Fig. 1).

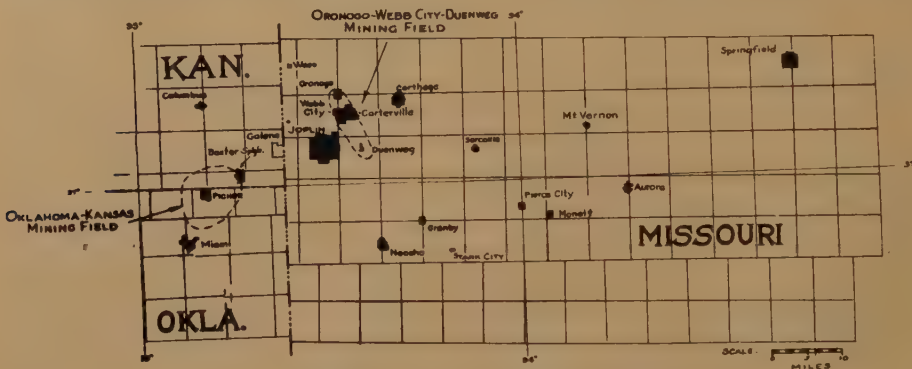

Fig. 1.—Index map Tri-State mining district.

Early Operations

Mining operations in what is now the Tri-State district date from about 1848 and have been continuous to the present. Apparently the first discoveries of ore were made about the same time near the present sites of Joplin, Oronogo, and Granby. Lead minerals were found at grass roots, probably in shattered rock in shallow excavations, then these minerals were followed in shear zones to deeper horizons. The mining of zinc followed. Before 1874, zinc ore was worthless and discarded as waste. According to published records, the total production of lead to that date was 49,000 tons of metal from 70,000 tons of concentrates.

Production

The aggregate ore production from the Tri-State district was recorded at intervals

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by Government and State agencies but continuous yearly records date from 1907.

The production from Jan. 1, 1908, to Jan. 1, 1943, according to U. S. Bureau of Mines figures, is shown in Table 1.

The greatest concentration of lead and zinc ores in the district was found around Picher, Okla., and Webb City, Mo. More than 75 per cent of the lead and zinc ore that has been mined to date came from

TABLE 1.—*Production in Tri-State District*, 1908–1943

	Tons	Tons Sphalerite Concentrates	Recovery in Concentrates	Tons Carbonaceous and Siliceous Concentrates	Tons Lead Concentrates	Recovery in Concentrates
Crude ore	304,763,098	14,063,405	4.61	306,334	2,338,875	0.77
Tailings	94,526,951	1,062,672	1.12		9,035	0.01

RECOVERABLE METALS, TONS

	Zinc	Lead
Tri-State district	7,899,429	1,825,005
United States	18,906,567	16,738,007
Percentage of production by the Tri-State district	41.78	10.9

Progressive improvements in milling practice during this 35-year period reduced the average tailing losses from several per cent in blende to the present low of about one per cent in blende. Lower grade tailings can be produced but the grinding cost is too high in comparison with the returns to warrant the expense.

about 20 sq. miles around Picher and about 10 sq. miles around Webb City. The remaining 25 per cent was produced from mineralized areas around Galena, Kansas; Waco, Kansas-Missouri; Granby, Aurora, and Wentworth, Missouri; and from innumerable smaller mineralized areas in every part of the Tri-State district.

Numerous papers have been published on the geology of the Tri-State district and the reader is referred to them for additional information regarding the subject.

The ore deposits of the Tri-State district occur chiefly in certain favorable beds in

TABLE 2.—*Geology and Ore Deposits in the Tri-State District*

System	Series	Character of Formations	Thickness, Ft.
	Pennsylvanian	Shale and sandstone (Cherokee)	0 to 250
	Mississippian	Chester—limestone, sandstone, and shale. Occurs as outliers, mainly in sink holes in Missouri. Overlies the Boone in the Oklahoma-Kansas mining field	0 to 100
Carboniferous		Boone—limestone, cottonrock-dolomite, and chert—originally all limestone	100 to 400
		Shale—Northview of Chattanooga. Absent in most of district	
Ordovician		Largely dolomite	700 to 1000
Cambrian		Dolomite and sandstone	0 to 800
Pre-Cambrian		Granite and probably other igneous rock that intruded the granite	

As shown in Table 1, nearly 100,000,000 tons of tailings have been re-treated and in some cases this process has been repeated two or three times. With present zinc prices, a blende recovery of 0.4 per cent or above pays the operating expenses.

the Boone formation (Fig. 2). In order of importance as ore reservoirs, the first four are: *M*, *K*, and *O* and *R*, the last two generally mined as a unit in the Oklahoma-Kansas mining field. Around Oronogo *O* and *R* beds are separated by about 20 ft.

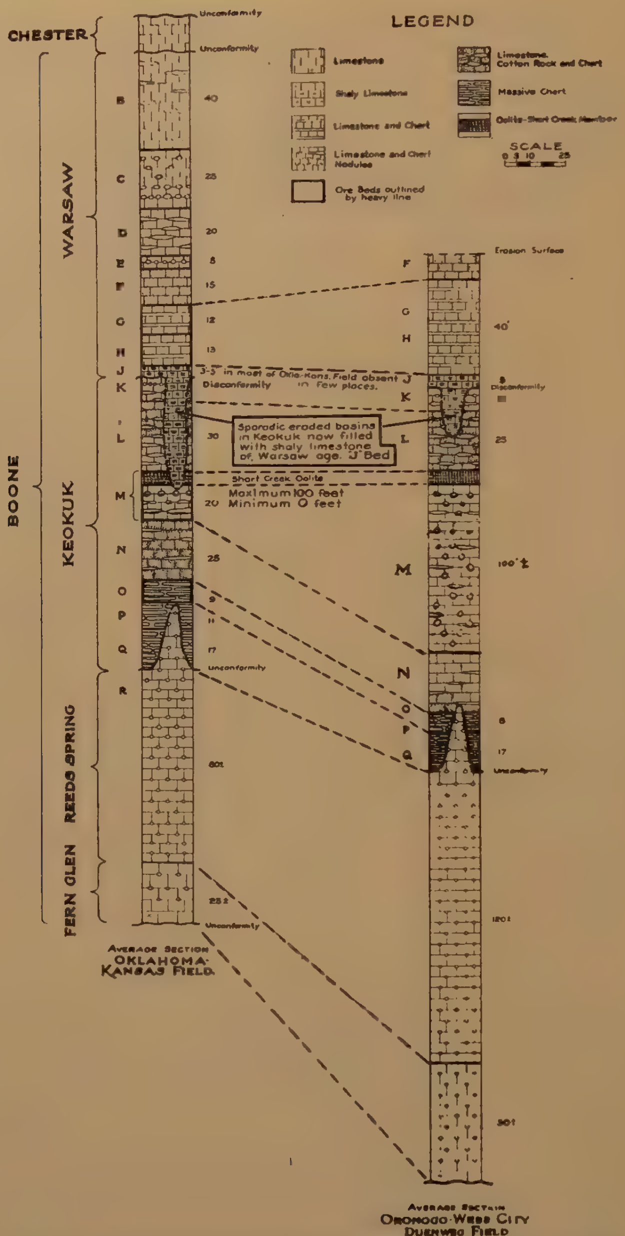

FIG. 2.—STRATIGRAPHIC SECTIONS OF THE ORE-BEARING HORIZONS.

of barren chert. All these beds are barren except in zones of premineral structural deformation.

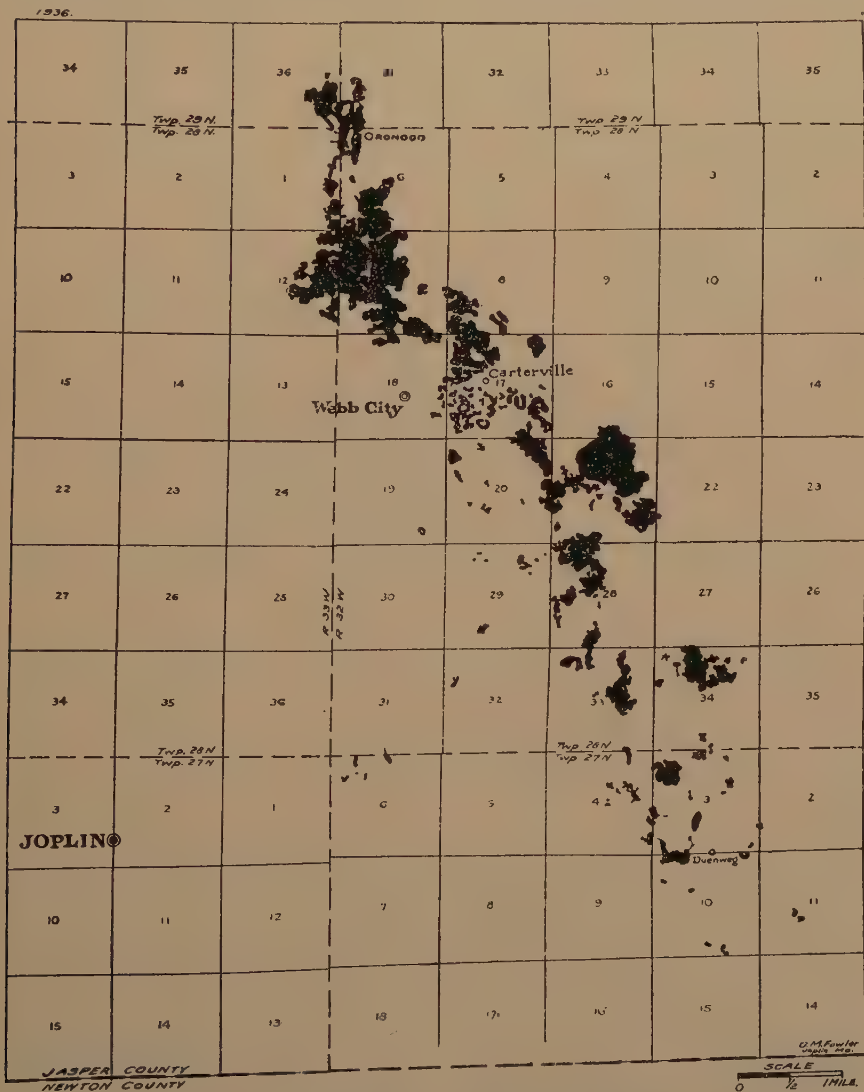

FIG. 3.—ORONOGO-WEBB CITY-DUENEVEG FIELD.

The subsurface structure of the area in the Tri-State district in which the ore bodies have been found may be likened to that of an oil field composed of numerous domes, basins, anticlines, synclines and allied flexures. This comparison is strikingly noticeable in the Oklahoma-Kansas mining field, where nearly all the structures are relatively small and differ greatly in both vertical and horizontal dimensions. In area they range from a fraction of an acre to many acres.

In a few of the major shear zones in the Oklahoma-Kansas field continuous ore has been mined from *R* bed to *E* bed or above.

(See Fig. 2.) However, in most places where the shear zones penetrate the strata only beds *G*, *H*, *K* *M*, and *O* contain ore. They are thinner bedded, contain many stylolites, and shattered more readily than the massive beds between them, which remained barren.

Nearly everywhere in the Tri-State district the chief ore horizon is *M* bed (Fig. 2). Its average thickness is 20 ft. in the main part of the Oklahoma-Kansas mining field. Erosion in Medial Mississippian times removed part to all of *K*, *L*, and *M* beds sporadically in the Tri-State district and deposited a greenish limy shale, *J* bed, which varies in thickness from a few inches to 40 ft. and is barren. In many places *J* bed rests directly on *N* bed. Eastward *M* bed gradually thickens to about 80 ft. around Baxter Springs, Kans.; 100 ft. around Joplin, Mo.; 145 ft. between Sarcoxie and Pierce City, Mo.; and 210 ft. around Springfield, Mo. It is ore-bearing from top to bottom only in shear zones in the Oklahoma-Kansas field and in some chimney ore bodies, such as the Oronogo Circle mine in Missouri. The continuity of *M* bed has been established from the Oklahoma-Kansas field eastward to about the vicinity of Mount Vernon, Mo., by examination of the cuttings from hundreds of churn-drill holes and by observations in many mines. Eastward from Mount Vernon is less conclusive because of fewer drill holes.

The present available information, based on sequence and characteristics of strata above and below it, indicates that *M* bed (Fig. 2) is the equivalent of the upper part of the Burlington member of the Mississippian formation. Westward from Springfield, Mo., this formation is obscure, probably because it thins gradually, and in parts of the region the limestone was chertified, flexed, shattered, and leached. *M* bed is the surface formation over a very large area in southwestern Missouri.

Widespread shattering of marked intensity occurred in many parts of the Tri-State district, particularly around Picher, Webb City, and the smaller areas mentioned above. Zones of intense deformation, where the deformed and shattered strata moved a few inches to several feet instead of many feet, formed particularly favorable ore reservoirs. Shear zones are numerous throughout the district and range from a few inches to several hundred feet in width and from a few feet to several miles in length.

Prospecting Methods

Early prospecting was done through shallow shafts, then, because of the water encountered, by churn drilling. The drilling was generally done in and near shale depressions or at random until ore was encountered, then the ore was delineated by additional drilling and mined through numerous shafts. The shafts were generally closely spaced because underground transportation was awkward and costly. With improved pumping equipment and underground transportation methods the underground workings were enlarged and deepened around central mining units.

Since 1928 much of the prospecting for ore in the region has been done, first, by drilling holes about ¼ mile apart, in order to determine the stratigraphy, mineralization, and other pertinent data. If conditions were favorable, additional holes were completed to delineate the ore bodies. The ore bodies range in size from a few cubic feet to many thousand cubic yards. Mining is done almost entirely in open stopes, which are supported by irregularly spaced pillars.

More than 50,000 churn drill holes have been completed in the Tri-State district to date. The cuttings and logs from this drilling have been studied in detail and utilized to the fullest extent in the search for ore. The drilling depth for ore ranges from a few feet to about 500 ft. Several

hundred holes, from 800 to 1200 ft. deep were drilled into or through the Rubidoux sandstone in order to tap the excellent water found there for civic and domestic resources are of inestimable value to the inhabitants in comparison with any conceivable tonnage of ore that may be found in or below them.

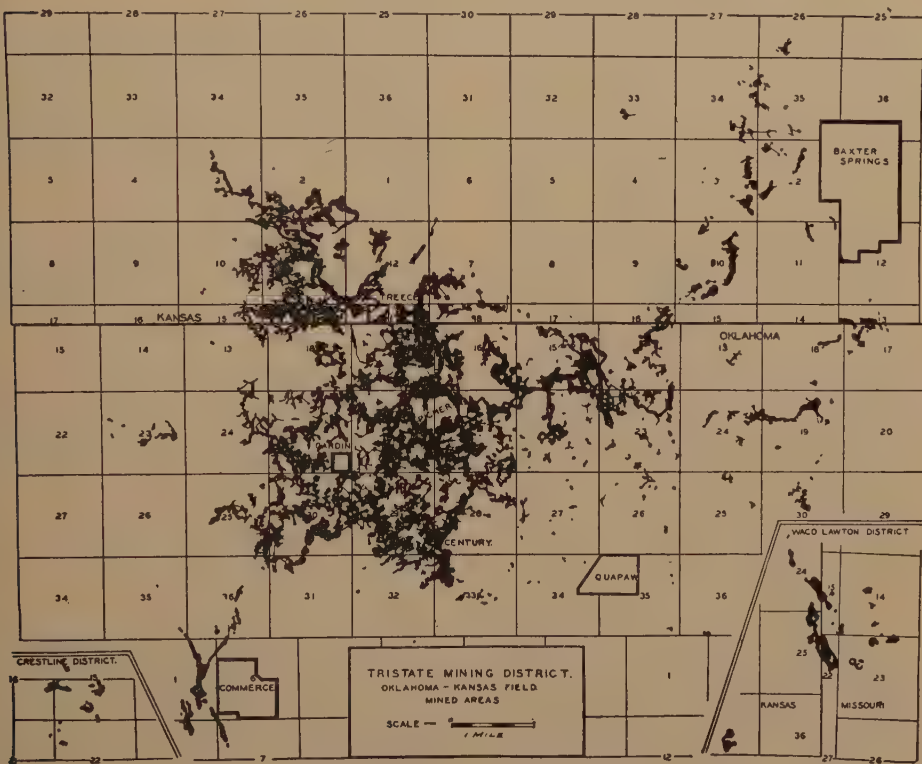

FIG. 4.—OKLAHOMA-KANSAS MINING FIELD.

purposes. Nearly all of these holes were barren below the known mining horizons. The granite basement that underlies the sedimentary series in the entire region at depths of 1200 to 2000 ft. was reached in a score of holes.

The work to date indicates little likelihood of finding ore below the Mississippian series. It is probable that ore will never be mined from the Rubidoux sandstone or below, even if found, because of the cost of controlling the large volumes of water known to be present in numerous sandstone members in these formations which are more than 100 ft. thick. Also, these water

All of the ore bodies in the Tri-State district localized in zones of regional premineral structural adjustment. Their shapes and dimensions were determined by the courses and intensities of the stresses involved in the process. The stresses were relieved by movement either vertical to the strata or parallel with them. The vertical movement that made ore reservoirs in shear zones and chimneys varies from a few inches to a few feet. The extremes in vertical deformation of strata in the Oklahoma-Kansas field are 350 ft. Data are lacking with which to measure the horizontal movement that opened

certain strata and made the reservoirs for the blanket or sheet-ground type of ore body. It seems evident that horizontal movement also formed the stylolites that are so prevalent in certain strata throughout the region.

It may be of interest to note the many structural similarities between the bedded deposits of the Tri-State district and the manto deposits of northern Mexico. However, the Mexican ore deposits and everything connected with them are generally on a much larger scale and the intense deformation is more concentrated. The bedded, transected strata and chimney types are common in both regions.

Mining

The mining depths range from grass roots to about 400 ft. The usual mining method throughout the district is open stopes where the roof is supported by pillars or legs spaced at irregular intervals. The stopes range in length to 1000 ft.; in height from 5 ft. to about 125 ft., and in width from a few feet to 100 ft. The pillar spacing is determined by the physical character of the formations. It is generally possible to locate them in leaner portions of the ore bodies, or in barren strata. As originally opened, about 10 per cent of the ore in the mines is in the pillars and legs and they are gradually reduced in size as needed to maintain production until 95 per cent or more of the ore in the property is won.

The mining height in the blanket or sheet-ground type of deposit varies from 5 ft. to 30 ft. Pillars are spaced as deemed necessary to support the roof, which generally is massive, thick-bedded chert.

Open-pit mining is done in special cases, particularly in chimney-like ore bodies, as at the Oronogo Circle mine, where the ore extends from grass roots to a known depth of about 300 ft. The present pit is about 800 ft. in diameter and 250 ft. deep. Many other chimney-like ore bodies from a few feet to several hundred feet in diameter have been found but nearly all of them are capped with barren strata that are from 100 to 300 ft. thick, and usually mined by open stope methods.

In the mining operations ore bodies less than 12 ft. high are drilled and blasted as a unit. Higher faces are mined on benches or steps, each 10 to 12 ft. high, which extend progressively forward from the floor to the top of the ore body.

Production at Woodchuck Mine

In order to show the intensity of zinc and lead mineralization where conditions were particularly favorable, the recorded production from the Woodchuck mine is given here, as follows:

Aggregate rock tons of crude ore milled	2,175,022
Aggregate zinc concentrates produced, tons	136,513
Metallic content, per cent	59
Aggregate lead concentrates produced, tons	28,534
Metallic content, per cent	82

This property, opened in 1915, comprises 40 acres in the SE ¼ NE ¼, sec. 30, T. 29 N., R. 23 E. The ore localized chiefly in shear zones and in some places its vertical limits exceeded 100 feet.

Underground Transportation

Mechanical loaders of various kinds are used very extensively underground. Hand shoveling is gradually being abandoned except in the smaller ore bodies or isolated properties.

The ore is transported from the headings to the shafts by extensive tail-rope haulage systems, electric motors, storage batteries on rails, and storage batteries on trucks with rubber tires. Mules are gradually disappearing except in the smaller properties.

Hoisting

Except at a few places, the ore is hoisted in steel cans with a net capacity of 1400

lb., at the rate of 80 to 100 cans per hour. At some larger units the shafts are equipped with cages and cars of 2 to 3 tons capacity or with skips of similar capacity.

Surface Transportation

Surface hauling from the mines to the concentrating plants is done largely in standard-gauge railroad cars and in trucks. The haulage distance varies from a few hundred yards to 50 miles, which is the distance from the Oronogo Circle mine, in Missouri, to the Eagle-Picher Central mill, in the southern part of the Oklahoma-Kansas field.

Future

Parts of the Tri-State district have been prospected or examined sufficiently to demonstrate that the important ore bodies therein have been found. Many hundred square miles remain to be prospected, and this is being done gradually by many mining companies and individuals.

The Tri-State district is one of several important mining districts of the Central States region. None of these districts have been adequately prospected and their combined area constitutes only a very small part of the region in which ore may be found. The lead and zinc ore deposits that have been developed in the region to date were localized largely by structural deformation. This condition is of marked similarity in the Tri-State and southeast Missouri lead districts.

The available evidence regarding all these ore deposits is being reappraised at intervals by many investigators. It is probable that their work will result in the discovery of important new ore deposits.

Part II. Milling

By R. E. Illidge

The fundamental scheme of milling Tri-State ores is gravity concentration supported by differential flotation. The economics involved have dictated, in no small degree, this choice of treatment. Another primary reason is the character of the ores. With gross mill-feed values of around $2 a ton, it is obvious that no elaborate system of treating total heads can be considered.

A means of early elimination of as large a fraction of mill heads as possible is the first essential concentrating step. This leaves an enriched product high enough in metal content for subsequent processing. In the entire history of Tri-State milling this fundamental principle has been recognized. The few who have tried to prove it fallacious never survived.

Significant improvement in milling practice, beyond doubt, will be the means to more efficiently and more economically accomplish the objective. It will take on more serious meaning as the more refractory portion of the district's ore bodies comes into production.

Missouri Section of Field

Milling practice was initiated and developed in the Missouri section of the field. Here very pure sphalerite and galena are deposited in horizontal bands of varying thickness. These bands are separated by mineral-free flint, chert, or occasionally dolomite. Crushing this type of ore to about ½ in. liberates much free mineral, which can be enriched by jigging into finished concentrates. The resultant middlings are small in tonnage and require only moderate roll grinding to liberate the locked minerals. Using jigs, tables and flotation cells, this type of ore responds beautifully, yielding high extractions and low tailing values.

Picher Area

The ores of the Picher area belong to the breccia type containing an abundance of jasperoid. When jasperoid represents a large fraction of the metal content, difficult problems present themselves. Since grains of gravity anywhere from pure gangue to pure mineral are present, it is appreciated that rougher jigging must be done on very small gravity differences. To effect maximum recovery means a low ratio of concentration. Because particle shape enters into the picture, it is next to impossible to make a low jig tail with this type of ore. A jasperoid concentrate is rich enough to withstand grinding costs down to point of maximum liberation, but the problem is to make a pure concentrate to ensure richness and at the same time keep each jasperoid grain out of the gravity tailing. Recoveries are lower on this type of ore.

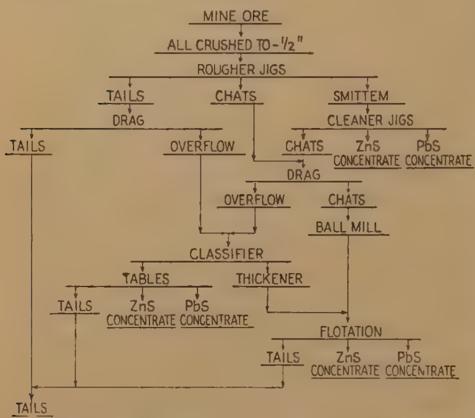

FIG. 5.—FLOWSHEET OF TYPICAL TRI-STATE MILL.

Ore-dressing Technique

Typical ore-dressing technique of the district incorporates four essential steps: preparation of the feed, primary gravity concentration, secondary gravity concentration, and slime concentration. Mine-run ore is grizzlied at 8 or 9 in., the oversize being hand-hammered through. Primary crushing is done by short-stroke, high-speed jaw crushers followed by slow-speed geared Cornish rolls. This product is screened on vibrating screens at about ½ in. After desliming in Esperanza type classifiers, the unsized feed is rougher-jigged, making three products: (1) an enriched product, which is cleaned into finished lead and zinc concentrates, (2) a middling for regrind and (3) a tailing for discard. The jig middlings and the tails from the cleaning jig are reground and treated over either tables or jigs, sometimes both. Treatment of middlings varies considerably, following many kinds of flowsheets, but all make finished concentrates and tailings. High-speed spaced rolls and ball mills are used in the middling circuits. Rod mills have never met with much approval. Differential flotation produces finished lead and zinc concentrates and a tailing for discard.

A flowsheet of a typical Tri-State mill is given in Fig. 5.

While not intending to give meticulous details of milling practice, it is worth while to say a little more about the rougher jig because that unit is the very heart of the district's typical mill. About 90 per cent of the mill feed is rougher-jigged. More than ¾ of its feed is discarded as a tailing, which contains about 10 per cent of the total values originally in the jig feed. The middlings contain about 5 per cent and the enriched fraction about 85 per cent. These values are concentrated into approximately 20 per cent of the weight fed to the jig. Extraction of the metal content to +65-mesh is very good, so this unit not only regulates metallurgical efficiency but regulates costs as well. Lowering the metal content of the tailing at this point has more merit than lowering it at any other place in the mill, but the process must be an economical one.

Central Milling

Though the district is almost a century old, only in recent years has the leasing

method of doing business been changed. Formerly, each landowner stipulated that his ore must be treated in a mill erected on his own land. This resulted in a multiplicity of small operations. Size of plant, capital investment, and operating costs had to be in proportion to the life of the mine. Soon the district was faced with a situation in which many properties still had ample ore reserves, others could see the end in sight if new ore bodies were not uncovered before long, while a few were merely cleaning up what was left. Those in the first and second groups could carry on economically while those in the third group found themselves unable to provide the mill with its regular daily tonnage. Naturally this increased costs. When costs crossed the line of profitable ore the only thing to do was to shut down. If the concentrate market improved later on there would be an operating margin for starting again; but eventually all properties would become members of the third group. Here then would be a large number of separate mines with individual ore reserves so small that they could not afford the cost of a mill. Collectively, they would represent the equivalent of a huge new ore body. Could not one good modern mill be erected and treat ores from these various properties, which still had limited tonnage?

In 1929 the Bird Dog mill of the Commerce Mining and Royalty Co. began to treat ores from different landowners, demonstrating the utility and soundness of the idea. Eagle-Picher followed three years later with its Central mill. This plant, up to Aug. 1, 1942, had treated over 17½ million tons of ore. All that was prophesied for central milling has come true, and more. It is now the very life blood of the district, permitting ore reserves to be depleted to the last ton. It is doubtful whether the old Joplin type of mill will ever again assume its former importance.

The Central mill is unique in several respects. First, it uses the so-called sink-and-float process. Second, it has minimized the use of jigs, and, third, it does not employ the use of tables. The general picture is as follows: The current feed rate is 500 tons per hour, from which approximately 30 tons of combined lead and zinc concentrates are made. Ore is hauled both by trucks and standard-gauge railroad trains. After entering the grounds of the mill, each load is weighed and dumped into subsurface storage hoppers, each capable of holding a complete shipment.

The ore is jaw-crushed and placed on an inclined conveyor belt, which transports it, after sampling, to a large storage hopper. The arrangement of weighing, unloading, primary crushing and sampling is such that the weight and head sample of each incoming ore parcel is very accurately determined; and further, it is impossible to mix one ore parcel with another until each is weighed and sampled separately. This permits accurate accounting for each ore contributor, no matter how many there may be.

The ore is then cone-crushed in closed circuit through coarse screens. This feed is again screened and separated into coarse, intermediate, and fine fractions. The coarse fraction, which is the largest, is treated by the sink-and-float process in two steps. The intermediate fraction is jigged. The fines are floated. A flowsheet is shown in Fig. 6.

In the first step of sink-and-float, a large portion of the mill feed is discarded as a tailing very low in metal values, leaving an enriched product, which is ground in rolls and screened to make two products. One joins the original intermediate fraction and is jigged. The other is submitted to the second step of sink-and-float, where a somewhat higher tailing is thrown away and a further enriched product saved, which, after ball-milling, is sent to flotation. Jigs and flotation cells make finished lead

and zinc concentrates for market and tailings for discard. These few remarks about the mill serve only to give an idea of what takes place in a fundamental sense; actually the mill is complicated. It represents an investment approaching two million dollars with buildings that cover acres, containing miles of wire, belts and spouts. Several hundred men are required to operate it around the clock, and its railroad yard is one of the busiest in the country.

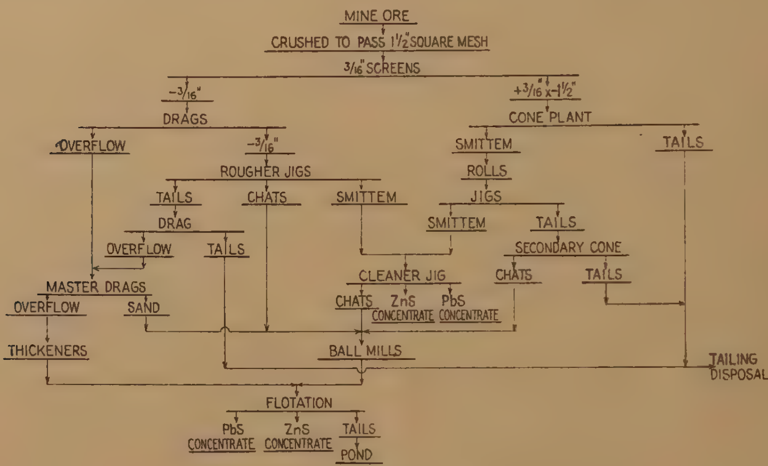

FIG. 6.—FLOWSHEET, CENTRAL MILL.

Sink-and-float Process

Two things are essential in the sink-and-float process—a container and a medium. The container is a large steel cone about 20 ft. in diameter and 24 ft. tall. The medium is a carefully controlled mixture of finely divided galena concentrate and water. The gravity 2.65 is chosen because of the characteristics of the Tri-State ores. As the specific gravity of jasperoid (quartz) is 2.65 and dolomite about 2.9, ore fragments lighter than 2.65 contain very little of value. If a continuous ore stream—ore that has been crushed to pass a 1½-in. screen and from which the sizes smaller than about 3/16 in. have been removed—is directed into the center of the cone-shaped container, which is kept continuously full of the medium of 2.65 sp. gr., within a few seconds after immersion of the ore fragments, the particles heavier than 2.65 sink to the lower end of the cone. A suitable air-lift device is provided to remove these particles from the cone. The lighter particles float on the surface of the medium and, because of the continuously fed ore stream, are crowded concentrically toward the periphery of the cone, over which they fall. Both products are now dripping with medium. Screens are suitably installed to permit medium drainage, which is put back into the cone. After natural drainage, the products are sprayed with water to remove the adhering particles of galena. This diluted medium is thickened, treated by flotation and eventually gets back into the cone at 2.65 sp. gr. More than half the mill feed is discarded in this first concentrating step. Sink-and-float is strictly a roughing device as applied to this ore dressing. It is the latest contribution toward the solution of the fundamental ore-dressing problem in this district. Gravity tailings have always contained significant amounts of blende. The obvious reason is that—with present knowledge—these amounts cannot be economically extracted to their irreducible limit. Postmin

eralization and premineralization gangue is substantially free of blende while the contemporaneous jasperoid can be made to liberate metallic content down to about 0.25 per cent if ground fine enough. When an economic roughing tail in the order of 0.25 per cent can be produced, the Tri-State district can start all over again.

Part III. Smelting Tri-State Concentrates

By B. M. O'Harra

Zinc Concentrates

Tri-State zinc concentrates are still largely treated, for the recovery of their zinc content, by the old established horizontal-retort process. One reason for this is that their high zinc content and low impurity and low precious metal content make them particularly amenable to treatment by this process. Another reason is that the necessary smelting capacity for Tri-State concentrates was already in existence before the development of the newer electrolytic, electrothermic, and vertical-retort processes, and the advantages of these newer processes have in general not been sufficient to justify the abandonment of horizontal-retort plants already in existence and the building of new plants.

Improvements in the horizontal-retort process in the last 10 to 20 years have been much greater than most metallurgists outside the zinc industry realize. They have consisted mostly of improvements in details of operating and metallurgical practice that are not spectacularly obvious, but which, nevertheless, have improved recoveries and costs to an extent that has enabled the horizontal-retort process to compete with the newer processes, whereas it might otherwise have been superseded.

At times considerable tonnages of Tri-State concentrates have been treated at the electrolytic plant at East St. Louis, and at vertical-retort plants, but the horizontal-retort plants still predominate in the treatment of these concentrates.

An expansion of smelting facilities such as has taken place in the past two years might, if it had been called for under more normal conditions, have taken to a greater extent the form of new plants utilizing the electrolytic, electrothermic, or vertical-retort processes, but under the conditions of war emergency, calling for bringing in the increased smelting capacity with maximum speed and minimum requirements of scarce materials and equipment, it has been preferable to expand capacity by rehabilitation or expansion of existing horizontal-retort plants.

Location of Smelters

A few of the smelters are owned by companies that now mine, or in the past have mined, much of the Tri-State ore. A few others are owned by important consumers of slab zinc. The remainder are owned by independent smelting companies.

Because of the large amount of fuel required by horizontal-retort plants, the location of these is largely determined by the location of the necessary supply of cheap fuel, or of markets for sulphuric acid that yield a profit from the latter, as a by-product of zinc smelting, sufficient to offset a higher cost of fuel. Freight rates on concentrates, and on slab zinc to market, are important but are subordinate to the factors mentioned above.

As a result of these considerations, the retort smelters treating Tri-State concentrates are found grouped into two classes: (1) smelters near supplies of low-cost natural gas, in Oklahoma and Arkansas, and (2) those near coal fields and convenient to markets for sulphuric acid, in Illinois, Pennsylvania, and West Virginia.

The markets for sulphuric acid accessible

to the Oklahoma and Arkansas smelters are limited, so, with one exception, these plants waste their roaster gases and do not produce sulphuric acid. At two natural-gas plants in the Texas Panhandle the freight situation with respect to Tri-State concentrates and zinc is unfavorable, so these plants do not treat concentrates from the district; they treat western and Mexican concentrates instead.

At smelters in Illinois, Pennsylvania and West Virginia, which use coal as fuel, the fuel cost is higher than where natural gas is used, but because the smelters are accessible to suitable sulphuric acid markets the fuel disadvantage is offset by the profit on by-product acid produced from roaster gases. Because of the necessity of having gas producers for converting coal to producer gas, of making provision for conserving fuel, and of having a sulphuric acid plant, the capital cost of the coal plants is much higher per unit of zinc produced than it is at the natural-gas plants.

There are three natural-gas smelters in Oklahoma and two in Arkansas, which ordinarily treat primarily Tri-State concentrates, though in the past couple of years two or three of these have treated considerable tonnages of western or foreign concentrates, and one of them now treats chiefly such concentrates.

There are three horizontal-retort smelters using coal as fuel in Illinois, two in Pennsylvania, and one in West Virginia, which ordinarily depend chiefly on Tri-State concentrates, but here again considerable tonnages of foreign concentrates have been treated in the past two years.

The total tonnage treated by the smelters using coal as fuel is considerably greater at present than that treated by the natural-gas smelters.

Equipment and Practice

Because of the difference in conditions at the two types of plants, there are certain differences in equipment and metallurgical practice, which are characteristic of the two types.

Because of the low cost of fuel at the plants using natural gas, any great investment for fuel conservation is not justified. The retort furnaces are of a more or less standardized type, from which the hot gases are wasted through the stack without any effort to recover their heat content.

These plants were built when Tri-State concentrates consisted almost entirely of coarse jig and table concentrates. Ropp roasters, which are mechanically rabbled, long, single-hearth, gas-fired furnaces, were used for dead-roasting such concentrates. Later, as milling practice in the Tri-State district changed and increasingly large tonnages of flotation concentrates were produced, most of these plants installed sintering plants and the Ropp roasters were used for preroasting followed by sintering, to convert the finely divided float concentrates to a suitable granular product for the most economical retorting. The Ropp roasters have high fuel consumption, and when roasting flotation concentrates cause a high dust loss as compared with a combination of multiple-hearth roasters (such as the Wedge, Herreshoff, and Skinner) with Cottrell dust precipitators. Their first cost is considerably less. As the natural-gas smelters already had the Ropp roasters, and to date the advantages of multiple-hearth roasters have not been considered sufficient to justify the capital cost of abandoning the Ropps and building new roaster plants, the use of Ropps is still universal at the natural-gas plants, with the one exception of the plant that produces sulphuric acid. The latter plant uses 12-hearth Herreshoff roasters, as Ropp roasters do not furnish sufficiently concentrated sulphur dioxide gas for manufacture of acid.

The smelters that use coal as fuel must convert it to producer gas for firing the retort furnaces. (Attempts to use pulver-

ized coal or oil for firing retort furnaces have been unsuccessful.) Generally they have some means for recovering the waste heat in the combustion gases from the retort furnace. There are two general ways of doing this: (1) to use regenerative furnaces with brick checkerwork regenerative chambers; (2) to use waste-heat boilers. The second method is the commonest. Both types of furnace differ somewhat in design from natural-gas furnaces because of differences in the properties of producer gas and natural gas.

As these smelters produce sulphuric acid, their roasting problem is quite different from that of the natural-gas smelters. Originally they all used Hegeler roasters, which for years were almost the only type of mechanically rabbled furnace in use in the United States for roasting zinc ores when sulphuric acid was produced as a by-product. These roasters have a series of superimposed longitudinal hearths and the concentrates progress from the top to the bottom hearth during roasting. The three lower hearths are muffled and fired with producer or other gas; the combustion gases thus do not mingle with the roaster gas. However, Hegeler roasters are not well adapted to roasting flotation concentrates, so when the Tri-State district began to produce large tonnages of such concentrates most of these smelters were forced to install roasters of the Wedge or Herreshoff type, which were used as pre-roasters preparatory to sintering. With them were installed Cottrell precipitators for recovering dust from the roaster gases.

Some of the old Hegeler roasters thus replaced were shut down and dismantled. Others are still in use for either pre-roasting or dead-roasting jig and table concentrates.

Recovery of Cadmium

The adoption of sintering as a step in the desulphurization of zinc concentrates made possible the recovery of cadmium as a by-product. Tri-State concentrates contain from 0.35 to 0.40 per cent cadmium. Much of this is volatilized in sintering, particularly if a little salt or other chloride is added to the charge for the sinter machine. The volatilized cadmium may be recovered in a Cottrell precipitator and the Cottrell fume treated for the production of metallic cadmium. This helps to defray the cost of roasting and sintering and now most of the smelters, using either natural gas or coal, catch the sinter-plant fume in Cottrell precipitators and are equipped to recover cadmium.

Natural Gas vs. Coal as Fuel

The economic comparison of smelting with natural gas and with coal as fuel naturally varies according to local conditions at different plants, but the fact that smelters of both types have continued to operate over a long period of years, through good times and bad, shows that they compete on a more or less even basis. Sometimes the natural-gas smelters may have an economic advantage over the coal-fired smelters, and at other times the competitive position is reversed.

Smelting Capacity

Because of the great demand for zinc during the World War of 1914–18, the zinc-smelting industry was greatly overbuilt at that time. Consequently, competition after the war was very keen. Many complete plants, and portions of other plants, were abandoned in the course of the years subsequent to the war. The plants that could do so naturally attempted to continue operation even at a small loss or small operating profit, disregarding return of capital, with the result that smelting margins were kept down by competition and, except for occasional periods of prosperity, zinc smelting was not on the whole a particularly profitable business. This was particularly true during the de-

pression of the thirties, until the beginning of the present war.

With the increased demand for zinc brought about again by war, and especially since the conquest of Belgium and France cut off smelting capacity, the excess of smelting capacity in the United States has given way to a deficiency. The consequent need for increased smelting capacity was met at first chiefly by rehabilitation of abandoned plants, or abandoned furnaces at operating plants, and later by building new furnaces at existing plants.

The return of peace will probably again find the zinc-smelting industry overbuilt and no doubt there will be another fight for survival of the fittest, with some plants forced to fall by the way. However, that is looking to the future, which is difficult to foresee with any accuracy now.

Uses of Zinc Concentrates

Though most Tri-State zinc concentrates are treated for the production of slab zinc, a considerable tonnage is used for the production of zinc oxide, both lead-free and leaded, by the well-known American or direct process, and smaller tonnages for the production of lithopone, zinc sulphate, and other zinc compounds.

Lead Concentrates

Although the Tri-State mines are predominantly zinc mines, they produce also a considerable quantity of lead, amounting over a period of years to approximately 10 per cent of the total lead production of the United States.

As the Eagle-Picher Mining and Smelting Co. is the largest producer of lead concentrates in the district, and as its lead smelter at Galena, Kans., is within a few miles of the field, the larger share of the lead concentrates from the district naturally tend to go to that plant for treatment. Smaller amounts go to smelters at Herculaneum, Mo., and Alton, Ill., and small amounts are used by pigment plants for the manufacture of leaded zinc oxide. (The Alton and Herculaneum smelters obtain the major part of their concentrate supply from the Southeast Missouri Region.)

Because of the high grade of the lead concentrates produced in the Tri-State field, averaging in the neighborhood of 80 per cent lead, they are particularly suited to smelting on ore hearths rather than by the usual western method of sintering and blast-furnace smelting, and both the Galena and Alton smelters use the ore-hearth process. Herculaneum has sintering machines and blast furnaces. The ore hearths make a direct recovery of most of the lead and the remainder goes into hearth "gray slag," which is then smelted in blast furnaces.

The lead produced from the Tri-State lead concentrates contains only a trace of silver or other impurities, hence no desilverization and very little refining is required.

Most of the lead produced at Galena is used at the Joplin plant of the Eagle-Picher Lead Co. for the production of litharge and red lead, lead pipe and plumbing supplies, and other products. Some is used at Galena for the production of lead silicate and lead pigments. An increasingly large proportion of the lead concentrates received at Galena is utilized for the direct production of basic sulphate white lead, blue lead, and leaded zinc oxide, without going through the preliminary stage of smelting to pig lead. This direct production of pigments from lead concentrates utilizes processes developed by the Eagle-Picher companies, and is made possible by the high purity of Tri-State concentrates.

Development of the Low-grade Manganese Ores of Cuba

By F. S. Norcross, Jr.,* Member A.I.M.E.

(New York Meeting, February 1940)

Manganese has long been considered one of the United States' most important strategic raw materials. Its indispensability in steel manufacture makes it vital to the nation's industrial life. Coupled with this indispensability is the fact that domestic producers have not been able as yet to supply more than a small part of the nation's manganese needs; consequently the United States has had to depend upon such distant foreign sources as the U.S.S.R., British India, the African Gold Coast and Brazil for the major portion of its ore.

The great bulk of the mineral is consumed in the form of ferromanganese, an alloy that averages 80 per cent manganese and 20 per cent mainly iron and carbon. Manganese is used as a deoxidizer and desulphurizer in steel manufacture. By combination with the residual oxygen and sulphur of the bath, it helps to produce a clean, sound metal. Approximately 14 lb. of manganese, it is estimated, is used in making one ton of steel.

For standard-grade ferromanganese, the ore should have a minimum of about 48 per cent Mn and preferably more. Beneficiation methods have not yet been developed for making the immense low-grade deposits in the United States salable at prevailing prices.

Dependence upon foreign sources thrust the United States into serious difficulties during the World War. At the start of hostilities in 1914, the nation was obtaining about four-fifths of its ore from the Eastern Hemisphere. Domestic production had been negligible, amounting to only 3000 to 7000 tons a year. As the war continued, shipments from the East, endangered by sea attack and blockade, dwindled rapidly. Imports from Brazil mounted. In this grave emergency, the United States, starting almost from scratch and under the stimulus of high prices, succeeded by 1918 in producing a peak of 311,000 tons, about 35 per cent of its consumption that year.

Since the war, the United States has depended upon foreign sources for more than nine-tenths of its high-grade ore. During 1936, 1937 and 1938, imports for consumption averaged 736,000 tons a year. During the same period, domestic shipments averaged 21,000 tons a year. Of the imports, the U.S.S.R. accounted for an annual average of 280,000 tons, the Gold Coast 208,000 tons, India 74,000 tons, and Brazil 72,500 tons.

But India and Brazil, which in 1936 ranked third and fourth in the list of importers, were displaced in 1937 and 1938 by a new and much nearer source—Cuba. From less than 1 per cent of United States imports in 1931, the island republic increased its shipments to more than 27 per cent in 1938. Cuba's average for this three-year period was 97,000 tons, and its total for 1938 was 131,000 tons.

History

The story of the economic development of manganese in Cuba dates back to the Spanish-American War days. John Greenway, then one of Theodore Roosevelt's

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Rough Riders, and later a prominent mining engineer, operator and capitalist of the Southwest, kicked a piece of ore that he recognized as manganese as he marched along a dusty Cuban road. He remarked to his companion, another Rough Rider, young David M. Goodrich, that some day Cuba might furnish manganese for American steel mills. During the World War, Cuba did provide a portion of the United States' desperately needed supply and, after the war, continued to ship a few thousand tons a year. This early sporadic manganese production was mainly from small high-grade deposits. Not until 1930 was an intensive effort made to develop a process that would permit the utilization of Cuba's low-grade ores.

David M. Goodrich, who became a colonel on Pershing's staff during the World War, and who is now chairman of the board of the B. F. Goodrich Company, continued to remember Greenway's remark. When the writer, who had been associated with Colonel Goodrich in the development of the Pecos mine in New Mexico, told him he thought a substantial deposit of low-grade manganese ore could be developed in Cuba and a metallurgical process worked out to concentrate the ore, Colonel Goodrich backed the project. After the initial experimental work indicated the feasibility of the project, the Freeport Sulphur Co. became interested in the venture. An investigation was made by Lindley C. Morton and Monro B. Lanier, and Freeport Sulphur Co. agreed to supply capital for experimental work and for the installation of the plant and equipment and the conduct of the business. This was arranged through the purchase by Freeport Sulphur Co. of stock in the Cuban-American Manganese Corporation, which owns all the stock of the Cuban Mining Co. The writer continued with the enterprise as general manager.

Many difficulties were encountered from the beginning and many new problems were presented. All the major ones have at length been solved by the operating management assisted by the technical staff of Freeport Sulphur Co., and also by many outside experts.

A number of the present operating staff have been connected with the project through all or most of its development. Frank Trotter, now general superintendent, has had charge of milling operations since their inception and has contributed largely to the many innovations. Brett Hurff, assistant mill superintendent, Charles Gardner, mine superintendent, J. H. Johns, mining engineer, H. F. Jewett, chief engineer and F. E. Wood, master mechanic, are also to be credited with many ingenious ideas and practical economies that had to be incorporated.

Construction of the plant, including dams for water supply, transport facilities and initial mining development, was completed in July 1932.

In the years following the completion of the plant, many natural hazards as well as technical and metallurgical difficulties were encountered. Three major floods, each greater than any in the 20 years that preceded the beginning of operations, caused great damage and partial shutdowns in successive years. A major earthquake and a revolution, followed by a revolutionary strike of workers all over the island, were additional setbacks.

The United States-Brazilian tariff agreement in 1935 cut the protective tariff on manganese by 50 per cent. As Cuban manganese enters duty free, this cut made uneconomical further operation at costs then current. The result was a shutdown of approximately a year until new improvements to cut operating costs were worked out. In all, the Freeport company has invested something more than $3,000,000 in this undertaking to develop Cuban manganese ores.

Cuban-American Manganese Corporation and its mining organization, Cuban

Mining Co., have been able to supply increasing amounts of ore to the American steel industry, as shown by import statistics for the United States. The company, furthermore, was awarded a contract by the U. S. Government to furnish the first 25,000 tons of manganese purchased under recently enacted Federal legislation to provide reserve stockpiles of strategic materials for national defense. In addition to assuring this nation a near-by source of high-grade manganese vital for national defense plans, development of the process in Cuba is significant for its possible application to domestic manganese deposits, which are of a grade similar to the Cuban ores.

Geology

The operating and reserve properties of the Cuban Mining Co. are situated within a district centering on the town of Cristo, Oriente Province, which lies in the northern foothills of the Sierra Maestra coast range, 10 miles inland by rail from the port of Santiago de Cuba. The principal operation at time of writing is the Quinto mine, near Cristo. Seven reserve ore bodies, readily accessible by rail, occur within a radius of 20 miles of this locality. Preparations are now under way for immediate operation of three of these reserve ore bodies.

The rocks associated with the deposits of manganese are Early Tertiary marine volcanic tuffs accompanied occasionally by limestones. These strata were warped and folded almost contemporaneously with deposition, and were later intruded by a granodiorite batholith, which now is exposed in the Sierra Maestra chain. Manganese mineralization in Oriente Province is of a regional character and probably had its inception in hydrothermal activity incidental to this intrusive epoch. Faulting attended and followed the mineralization.

In the Santiago-Cristo district, the principal ore bodies occur as concentrations of manganese oxides and hydroxides, and represent the selective replacement of the primary fragmental components of tuff beds that lie immediately below capping limestones. At times the base of the limestone itself has been metasomatized; and frequently irregular replacement masses of jasperoid chert, known locally as "bayate," are associated with the deposits and carry veins of high-grade manganese oxides. Ores in limestone and bayate are usually of good grade, but invariably are localized in small, irregular bodies. On the other hand, those resulting from the replacement of tuffs, though always low in grade, may form deposits of large tonnage. It is only in the Santiago-Cristo district that deposits of the latter type predominate. An ideal example of this class of ore body is the Quinto mine.

At this mine, two roughly stratiform layers of tuff ore are separated by a variable thickness of chloritized tuff, and underlie a series of clayey limestones. Bayate occurs prominently below each of these ore beds as thick, irregular stringers; and also locally cuts off the ore itself in large replacement masses, particularly along fault and shear zones. In restricted areas, the ore may merge laterally or vertically into unreplaced tuff. The dip of the ore body and associated strata is 15° to the north; but local irregularities are greatly increased by the presence of broad asymmetrical folds, trending east-west, and having dips on the northern limb approaching 90°; and by several fault systems, both normal and reverse in character, with throws up to 50 feet.

The structural irregularity of the ore body cannot be overemphasized. The rocks involved in the main folding have also been cross-folded, with the result that sudden reversals of pitch are common, and unpredictable changes occur in the direction and magnitude of dip. Fold crests may be broken and overthrust, and further complications are added by the groups of strike,

dip, and oblique faults, by the steep vertical structures due to folding and drag, and by the presence of large blocks of unreplaced tuff and sheared bayate.

The tuff ore of the Quinto mine has a grade of from 13 to 26 per cent metallic manganese, the ore extracted at present averaging 17 to 18 per cent. Limestone ore, which is not present in the Quinto deposit, ordinarily ranges between 25 and 45 per cent; while the manganese oxides associated with bayate will commonly give analyses between 30 and 55 per cent or higher.

In summary, the fact most to be stressed is the unpredictable variability of both ore and gangue. The ore exhibits rapid and marked horizontal and vertical changes in texture, mineral composition, grade, specific gravity, and hardness. Similarly, the nature and composition of the gangue is distinctly erratic, particularly with regard to its zeolite, bayate, clay, carbonate, and tuff content, its variable degree of cementation, and its range in specific gravity and hardness.

Mining

Among the ore bodies, the principal ones in either the operating or development stages at present are the Quinto, Ponupo, Sultana, and Abundancia mines.

Quinto Mine

The Quinto is the principal operating mine. It produces 1000 to 1200 tons of 18 to 20 per cent Mn ore per 8-hr. day from an open pit, which is now 2600 ft. long by 1000 ft. wide (Fig. 1). Two geological conditions dictate the type of mining equipment and the open-pit mining methods employed.

First, the ore body outcrops to the south and dips under increasingly heavy overburden to the north and west. The effect of this factor is shown by the rapid rise in recent years of the ratio between strip yardage handled and tons of ore mined; i.e., in 1933 the ratio was 0.72; in 1934, it was 0.52; in 1938, it was 3.13 and in 1939 (for about 10 months) it was 3.83.

Second, the ore formation is highly irregular. This has forced the transition from the shovel to the more flexible dragline as the main excavating unit. The dragline is now employed for mining and loading the ore as well as for stripping the overburden.

Original stripping was by train haulage to spoil dumps, but this method has been superseded by the use of a Monighan walking dragline, having a 4-cu. yd. bucket and 125-ft. boom, and tractor-scraper units that haul the dragline spoil outside of ore limits. The Le Tourneau 12-yd. scrapers, a recent innovation, are also employed in conjunction with a bulldozer and rooter for stripping solid overburden.

Ore is loaded by 2¼-cu. yd. dragline and a 2-cu. yd. shovel into 7-ton dump cars, which are hauled to mill in 9-car trains by gear-driven steam locomotives. Studies are now in progress to replace present haulage with motorized Diesel system. The mill is 1⅛ miles from the mine.

The overburden requires little blasting; where necessary, jackhammer holes are shot with 40 and 60 per cent dynamites. The ore is blasted with lines of 14-ft. jackhammer holes loaded with either 60 per cent dynamite or "Rompe Roca," a locally made stumping powder.

Ordinarily, open-pit mining shows a regularity of cuts and benches that allow consistent excavation, sampling and orderly procedure. This, however, is not true at Quinto. The Main Upper ore body resembles a warped plane or surface. It dips generally and irregularly to the north with many transverse and irregular dips. Some dips are short and sharp, and in places 2 to 10-ft. sharp faults are encountered. This upper ore body varies in thickness from 10 to 40 ft., and includes waste or low-grade sections or slices within it, similar in shape

to large pancakes, sometimes 200 ft. long. The mineral content of the upper and lower portions of this bed usually varies in metallic content and characteristics. Often solid bayate masses occur irregularly in the beds. A level shovel cut or bench, therefore, may be in the upper portion one day, then pass successively through part of the upper ore section and middle waste and lower ore section, and back again. The ore delivered from the same shovel cut may change two or three times daily. Operators of electric shovels and draglines have become sufficiently skillful to "shovel-sort" the "faces," thereby eliminating intermediate waste and low grade.

Fig. 1.—Entrance to Quinto pit.

Underlying the Main Upper ore body are two Lower mineral horizons or beds, which are even more irregular than the upper bed.

Original churn drilling on 100 and 200-ft. centers, although disclosing the general rolling character, failed to give a correct picture because of the many minor changes that occurred in short distances of 20 to 40 ft. It soon became apparent that regular shovel cuts and benches either in mining or stripping operations were not efficient, and gradually all work, including car loading, was performed by draglines. As a result of these condition and because original tonnage and grade estimates eliminated enclosed waste and low-grade areas, as well as small isolated ore areas of good grade, the manipulation of the shovels and draglines in selective mining has resulted in regular increases in tonnage and grade recovered from all calculated areas since operations began.

In working the underlying areas after the Upper ore body in any section has been stripped and removed, it is often the practice to strip the intervening waste with draglines, casting waste for two or three days, and then mining stripped ore and recasting waste into ore excavation, and proceeding until the smaller ore areas are removed. This eliminates expensive hauling of waste from pit, but does leave a rather messy working appearance in such areas.

As the overburden increases to the north, and as the original pit increases in size, it

is planned to excavate these deep sections first and then use these areas for dump room for the southern and easterly extensions of the beds, thereby reducing waste haulage, which is very expensive, especially during the spring and fall rainy seasons.

In general, now that the pit area is affording sufficient room, the future scheme will approximate that of dredging a placer deposit, where the dredged areas are filled in behind the dredging, thus reducing the excess overburden which is now being hauled from pit by 12-yd. Le Tourneau scrapers.

Although strip operations in point of yards handled are now more than five times as great as in the beginning, costs have only increased 30 per cent owing to change in methods and the use of larger and more flexible equipment.

If the same "bedding" methods or principles used at custom smelters for lead and silver ores could be applied, many of the high and low spots of ore irregularity could be eliminated, and milling would be simplified. For various reasons, this scheme was not economically practical.

Ponupo Mine

The Ponupo mine, 11 miles by rail from Quinto, is being prepared for open-pit mining operations. A second Monighan dragline, swinging a 5-cu. yd. bucket on a 120-ft. boom, is stripping the capping limestone, which is drilled for blasting by both electric blast-hole drills and heavy drifter drills placed on wagon mountings.

The ore is to be mined by a 1½-yd. shovel loading into either 16 or 18-yd. wagons hauled by Diesel tractors to ore-delivery points.

Sultana and Abundancia Mines

The Sultana and Abundancia mines are being developed by churn-drill holes placed on 25 to 100-ft. centers. Four churn drills are in operation, developing and exploring these areas.

Character of Ore, Gangue and Water

The ore is chiefly characterized by extreme variability in physical structure and chemical composition, an anomaly no doubt partly accounted for by the fact that the mechanics of replacement were affected by the heterogeneous nature of the tuff. Table 1 illustrates the range in specific gravity, hardness and composition of the various minerals. The only manganese minerals in the ore definitely identified to date are the common oxides and hydroxides: pyrolusite, psilomelane, less commonly manganite, and rarely wad. No carbonates and sulphides of manganese have been observed, even in samples taken of ore lying 300 ft. below the surface.

TABLE 1.—*Character of Minerals*

Mineral	Hardness	Specific Gravity	Constituents
Wad.........	0–6	2.0–4.6	MnO_2, $MnO{\cdot}H_2O$
Pyrolusite.....	2–2.5	4.7–4.8	MnO_2
Manganite....	4	4.2–4.4	$Mn_2O_3{\cdot}H_2O$
Psilomelane...	5–6	3.7–4.7	H_4MnO_5
(?)Braunite...	6–6.5	4.7–4.8	$3Mn_2O_3$, $MnSiO_3$

The various oxides exist either in a pure state or tend to intermix in all possible proportions to form complex aggregates. Because of this, figures for specific gravity and hardness range between the maximum and minimum limits of the end minerals in the group. The variations in chemical composition may also be ascribed in part to this phenomenon.

The process of replacement is at times highly selective. In some instances, only one element of the tuff, such as the cementing matrix, will be replaced. At other times the process is all-inclusive. There is thus a wide range in particle size, with lower limit below minus 600 mesh.

Variations in the gangue are common. They are due principally to: (1) differential replacement of tuff constituents, such that one block of ore will contain a different suite of unreplaced rock minerals from that in an adjacent block (chief component minerals are lithic fragments, ash, crystals of feldspar, quartz, etc.); (2) differences in character and amount of introduced substances—bayate, quartz, zeolites, calcite, pyrite; and (3) differences in character of hydrothermal alteration products—chlorite, sericite, iron oxide, clay material, and so forth.

All ores contain more or less soluble salts, present in the form of carbonates. The concentration varies with the difference in bed structures. The differences in soluble salts, specific gravity and hardness of both the mineral and gangue are responsible for the difficulties encountered in the milling operation.

Development of Treatment Process

After three months preliminary investigation, the following general and tentative flowsheet appeared to offer best possibilities for a successful treatment: (1) jigging, (2) tabling, (3) flotation (rejects and middlings), (4) sintering (table and flotation concentrates). Extensive pilot-mill tests and experimentation eliminated tabling and indicated a doubtful value for jigging, and as all flotation continued to give satisfactory and improving results, that process was finally adopted.

Sintering tests were made with a Dwight-Lloyd machine as well as by an F. L. Smidth and Co. sintering kiln. Smidth and Co. at that time reported failure to make nodules without adding silica, and the D. and L. sintering machine was selected.

Intensive metallurgical tests were conducted for many months. Carloads of samples were taken extremely carefully as representative of all sections, including deep shallow and surface ores. River waters from flood and dry seasons were tested and impregnated with arbitrarily selected plant growth. Additional factors were brought to light even after the main plant began operations.

The unusually marked variations in the ore beds within very short distances described under Geology indicate reasons for many of the difficulties encountered.

The first plant flowsheet finally selected was as follows: (1) fine grinding (ball mills), (2) all flotation, (3) thickening and filtering, (4) Dwight and Lloyd sintering. The present operating flowsheet (Fig. 2) is: (1) jigging (with rejects to flotation), (2) fine grinding (rod mills), (3) flotation, (4) dewatering drags, (5) Smidth rotary nodulizing kiln. The general reasons for the process changes are given in the following paragraphs in the approximate order of their occurrence.

Settling.—One of the most pronounced characteristics of the mineral was the general rapidity of settling and its marked individual variations. This peculiarity was not discovered in preliminary laboratory and pilot-mill tests nor by the technical departments of various suppliers of equipment. Pilot tests were not of sufficient duration to disclose the "building up" effects of the settling, and as all pumps happened to be direct-gravity feed they did not clog at intakes during tests.

Shortly after work was begun, this difficulty became apparent, and it affected operations to such an extent that exhaustive tests and studies by specialists were immediately carried out to remove the trouble.

Ore of 29.3 per cent Mn content settled with eight times the rapidity of a 42 per cent ore; 14 per cent and 22.5 per cent Mn samples settled in almost equal periods. Of two samples of 30 per cent ore, one settled almost twice as rapidly as the other. There seemed to be no limit to the variability of settling rates, and the general rate was extremely rapid.

It was also apparent that the mineral, especially of the finer sizes, showed more or less cohesion between particles and acted like iron filings under a magnet.

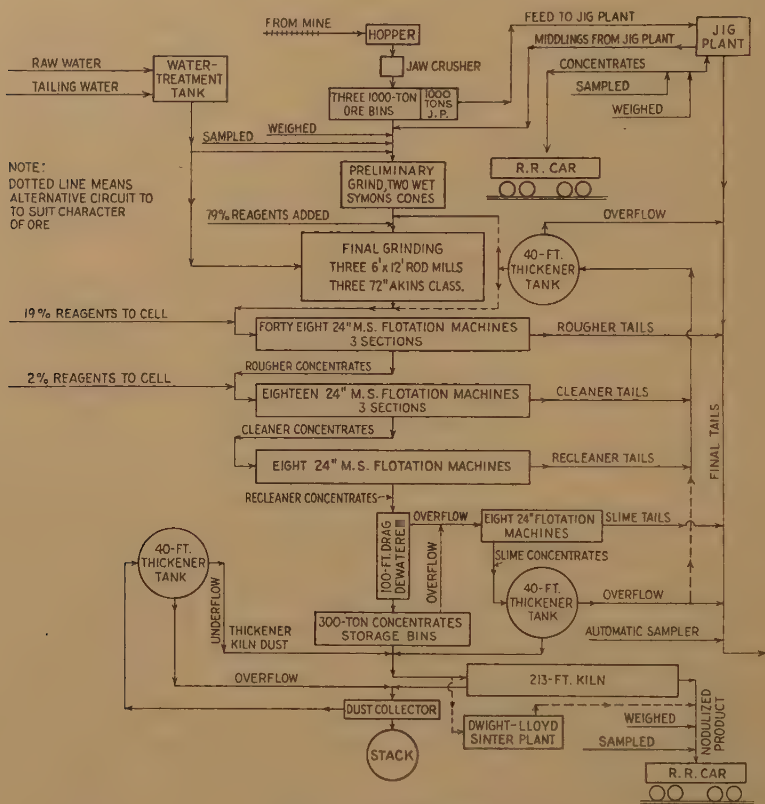

Fig. 2.—Flowsheet.

Colloid solutions showed more or less characteristic electric behavior. Varying specific gravities, together with marked changes in the percentage of slimes delivered from the grinding section due to ore variations, all contributed to the unfavorable settling condition and prevented efficient treatment.

Pump runners blocked, as did pump lines and intakes, launders, small pulp passages and dead corners in flotation cells, elevator boots, and similar places. The principal difficulty was encountered in thickener tanks, which constantly plugged up at all discharge points, and in filters that would not function.

It was decided to take advantage of the rapid settling rate, therefore the plant was shut down, thickeners changed, filters pulled out and a 6 by 100-ft. dewatering drag was installed (Fig. 3). This drag worked out of a pool into which all flotation concentrates were fed, and discharged into locally designed mechanical storage bins. These bins were air-treated and the water in concentrates drained off at the top. Thirty minutes to one hour was required for drying a 50-ton bin sufficiently to screw-feed the concentrates to the sinter plant. Further improvement became effective

when rod mills were substituted for ball mills and "grind" became coarser and more consistent.

Belts.—The oil and fatty acid reagents caused belts to curl and disintegrate. *Remedies:* Use of small drag conveyors and the installation of the main elevator in the grinding section ahead of rod mills, to avoid handling of pulp impregnated with reagents.

Cattermole.—Excess slimed mineral plus oily and soapy reagents caused cattermole to build up in pipe lines, classifier blades, ball-mill discharge grates and pulp flow lines in flotation cells, pump runners, etc. *Remedies:* (1) elimination of ball-mill grates and altered reagent application; (2) replacement of ball mills by rod mills, which reduced the amount of mineral slimes; (3) further changes were made in amounts and points of application of reagents.

Flotation Froth Handling.—Considerable difficulty was experienced in pumping scavenger cell froth from scavenger flotation circuit on the lower level up to rougher cells. *Remedy:* Scavenger and rougher sections were all placed on the same level.

Sliming.—As a rule the manganese minerals are soft and friable. The ball-mill discharge ran as high as 70 per cent minus 200-mesh. This material was easily recovered in flotation, but was difficult to filter and would not all settle out in the dewatering drag conveyor, which had replaced the filters. Losses were unusually high for this reason. *Remedy:* replacement of ball mill by rod mills.

Flotation Difficulties.—On account of varying mineral characteristics, shape, hardness, size, specific gravity, etc., flotation cells treated a varying feed under fixed conditions. *Remedy:* rod-mill grind, which gave a much more uniform product, also automatic classifier control, solution density control, and capacity flexibility obtained by increasing capacity of various working sections.

Circulating Load.—All section by-products, such as cleaner and re-cleaner flotation tails, thickener overflows (graduated), dewatering drag overflow, were returned

FIG. 3.—DEWATERING DRAG CONVEYOR.

to rougher flotation circuit. At times no difficulty whatever was caused by this return building up; at others, it upset the whole mill. *Remedy:* installation of second thickening section for circulating load and additional flotation cells. Part of this re-treatment product of the circulating load was returned to rod mills and part to separate flotation cells, depending on the class of ore being treated.

These were the most prominent factors of inefficiency from 1932 to 1935, although other vital points of weakness had become apparent, such as: water conditions, soluble salts in the ore, dust losses, slime mineral losses, volatilization losses, reagent control.

While the plant was shut down after the United States-Brazilian trade agreement, the intensive research program under way was continued and by Jan. 1, 1936 solutions of these additional difficulties were indicated, as follows:

Water Treatment and Soluble Salts.—Test work had previously demonstrated that improved flotation was obtainable with distilled water or softened water. The water being used was not only impure but varied in hardness over the year, month, and sometimes by the day. Sanitation laws passed in 1935 made it obligatory to re-use tailings water. Soluble salts present in the mine ores occurred, moreover, without any

regularity or in any particular ore section. *Remedy:* a water-treatment plant, installed in 1936, to provide a consistent as well as a "softer" water.

The remedy for all three classes of losses was found in the installation, in 1936, of the F. L. Smidth metallurgical kiln, in use at that time in only two plants—two kilns

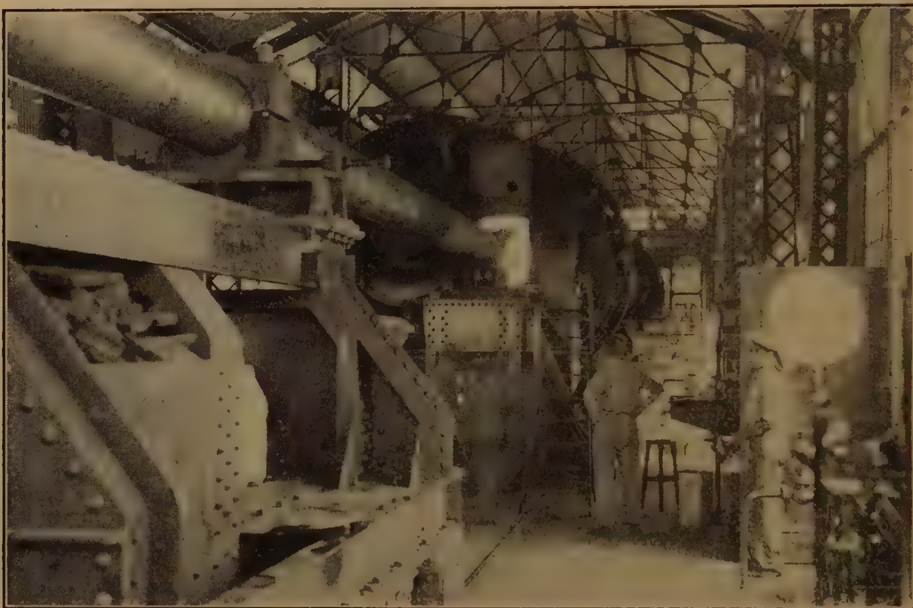

FIG. 4.—NODULIZING KILN, SHOWING SCRAPER BAR.

at Trinez, Czechoslovakia, and two in Luxemburg. This kiln is a rotary oil-fired unit similar to a cement kiln, but it has an enlarged zone 17 ft. from the discharge end of the kiln, which delimits the sintering zone to last 17 ft. where a mechanical scraper bar can reach and clean the walls (Fig. 4). The oil firing eliminated volatilization. The mixing of all coarse sand flotation concentrates with the mineral slimes from the various thickeners, making a 30 per cent moisture pulp feed to kiln, eliminated the majority of the dust losses and previous slime mineral losses, to say nothing of simplifying the operating flowsheet and reducing operating costs.

Losses.—Dust losses from the sinter plant had been prevalent from the beginning in spite of a special dust-collecting plant. An exact or even close metallurgical balance had never been obtained. Exhaustive tests showed that volatilization was taking place in the sinter operation. The fine mineral was in close contact with incandescent carbon, which caused volatilization and marked losses.

Losses of slime mineral were still occurring in the mill from tanks, drag dewatering conveyor and flotation concentrate bins. In 1934, to correct this, an oil-fired drier was designed locally and installed to treat the slime products after the largest part had been collected in thickeners. Thickener overflow losses were still present and the thickening and drier costs were excessive.

Reagent Control.—Continual variations in all the materials, such as ores and waters (even after the water-treatment plant was installed) served to exaggerate the usual

human errors of judgment and execution. Probably there were very few periods in a day when the correct amounts of reagents were being applied for best results, to say nothing of wastage. It was decided to eliminate the human element as far as possible, and for 18 months the staff worked on this problem. The installation of such a scheme was the last and latest improvement. All reagents and water are now automatically controlled, and are synchronized with the ore feed at the head of the mill.

General Results.—The mill crew, which on the first two boatloads of sinter shipped in 1932 consisted of more than 100 men per shift, now consists of 21 men per shift, and delivers three times as much product because of increased efficiencies.

Milling

Coarse Crusher.—The ore from the mine is dumped into a 120-ton bin and fed over seven 48 by 15-in. grizzly rolls by a 11 ft. by 48-in. super-duty apron feeder. The oversize is crushed by a 36 by 48-in. jaw crusher, to minus 4-in. ring size. Lumps of ore too large to pass the crusher opening are broken with a pneumatic rock breaker. The grizzly undersize and crushed material is conveyed to three 1000-ton cylindrical steel storage bins.

Fine Grinding.—The ore is fed by variable-speed 24-in. apron feeders from the storage bins onto a belt conveyor equipped with a Merrick weightometer and automatic sampler. After sampling the ore passes into a 4-ft. standard Symons cone crusher where it is crushed to minus ⅜ in. A 36-in. by 50-ft. bucket elevator carries the material to a 4 ft. short-head Symons cone crusher set to crush to minus ¼ in. Head room has been left to install a vibrating screen to choke-feed the short-head crusher. The ore contains considerable clay and it is necessary to add water to the cone crusher to prevent choking. The amount of water added is varied according to the clay content of the ore.

The minus ¼-in. product from the short-head cone crusher is fed by means of launders and a distributor box to three 12-ft. by 6-ft. Marcy rod mills in closed circuit with three 72-in. Akins classifiers.

Water Control.—Individual automatic density controllers regulate the amount of water added to the classifiers and maintain a constant pulp density of the classifier overflow. They are in a control room. The fineness of the grind can be changed by adjusting the density controllers. Recording meters in the classifier motor circuit, calibrated to record in tons of circulating load, are in the control room; also rheostats governing the speed of the apron feeders under the ore bins. By observing the amount of circulating load recorded by the classifier meters, an operator in the control room can determine the need for raising or lowering the tonnage of mill feed, thus keeping the mill circuits grinding at maximum capacity and compensating for the variation in the hardness of the ore, bin segregation, and other factors.

Reagents.—The reagents used are soap, gas oil, quebracho and lime. About 79 per cent of the reagents is added to the rod-mill feed and 19 per cent to the classifier overflow; the remaining 2 per cent is added at the head of the cleaner cells. The reagent feeders are the motor-driven tilting bucket-elevator type, commonly known as the Sterns-Roger reagent feeder. The amount of reagents fed is dependent on the speed of the feeder and the amount of tilt given the buckets when discharging.

Reagent Control.—The amount of reagents added per ton of ore is controlled by the amount of tilt of the reagent bucket. By synchronizing the motors driving the reagent feeders with the integrating disk on the weightometer, the number of buckets of reagents dumped is in direct proportion to the amount of ore being fed. Adjustments of the tonnage of crude ore fed are automatically followed by the reagent feeders. This method of feeding reagents

lowers the consumption of reagents and raises the recovery of minerals. The liquid levels of the reagent tanks are kept constant by an arrangement of a photoelectric cell and a solenoid valve.

In the control room are Brown liquid-level recorders connected with the reagent-storage tanks, and also a Merrick ratograph. The former indicate pounds of reagents added per ton of ore and the latter shows the ore tonnage.

Flotation.—The Akins classifier overflow is pumped through a distributor to three 16-cell sections of 24-in. all-metal sub-aereated Minerals Separation flotation machines used as rougher cells. A rougher concentrate is removed from the first six cells. The remaining 10 cells are used as scavengers and the froth is circulated to the head cell.

The rougher concentrate from each section passes through a Richard-Janney hydraulic classifier and the coarse, heavy mineral is taken out as a final product. This mineral is very difficult to float on the cleaner and recleaner machines, and if not removed builds up as a heavy circulating load in the mill circuit. The overflow from the hydraulic classifier is refloated in three 6-cell sections of 24-in. flotation machines. The concentrates are recleaned in an 8-cell flotation-machine section.

The recleaner concentrates, together with the hydraulic spigot product, are settled in a 6 by 100-ft. chain drag classifier. The overflow from the drag classifier is pumped to a 9-cell flotation section, making a tail to waste. The slime concentrate from this section is thickened and fed to the kiln. The deslimed mineral or sands from the drag classifier are stored in 50-ton steel bins that have screw feeders on the bottom.

After a tank is filled with wet concentrates compressed air is blown through the concentrates for 5 to 10 min. to agitate them thoroughly. The tank is then allowed to settle and the slimes are decanted. By blowing and settling the moisture is reduced from approximately 40 per cent to 18 to 23 per cent, which is low in view of the porosity of the mineral.

Circulating Loads.—The cleaner and recleaner tails and at times the overflow from the slime-concentrates thickening tank are thickened in a 40-ft. thickener tank. The thickened product is fed either to the grinding circuit or to the head of the flotation machines, depending on the character of the ore being treated. The thickener overflow is low enough in manganese values to be thrown to waste. One of the functions of the thickener tank is to prevent surging in the flotation circuit. Thickening the circulating load gives better operation and also enables the operator to estimate more accurately the amount of circulating load.

Mill Products.—Three mill products are made: a mineral sand from the chain drag classifier, a mineral slime from the drag-classifier overflow and a final tailing. The two mineral products contain approximately 44 per cent Mn. The ratio of mineral sands to slime mineral varies. Ores containing a large proportion of hard mineral give large amounts of sands and low slimes. The reverse is true of soft ores. The equipment is large enough to handle the extreme limits of the ores treated.

Kiln Feed.—The storage capacity of the concentrate bins and of the slime-concentrate thickener tanks is large enough to smooth out abnormal variations in the ore. This is done by mixing both products in a mixing drag in proportion, to give a constant moisture content. The mixing is done in a wooden tank 21 by 6 by 6 ft. A continuous chain drag driven at approximately 100 ft. per minute keeps the mineral slurry agitated. This tank also serves as a surge tank to compensate inequalities in the feed between the kiln and the mill-concentrate storage. A recording meter attached to the motor driving the mixing drag records approximately the moisture content of the slurry. A small variable-speed drag conveyor feeds the slurry to the kiln. The

amount fed is governed by the speed. The voltage from a small magneto attached to the drive motor records the speed of the conveyor.

Kiln Operation.—The fineness of the flotation concentrates necessitates nodulizing before shipment. This is done in a continuous kiln, 213 ft. by 9-ft. dia., built by F. L. Smidth and Co. The kiln consists of three zones; namely, drying zone, preheating zone and sintering zone. The preheating zone is larger in diameter than the others. This enlarged zone tends to limit the sintering to the sintering zone. A water-cooled bar with a cutter attached is mounted on a motor-driven car. This bar can be passed into the sintering zone and any sintered material that has ringed on the walls can be cut out (Fig. 4).

The kiln is oil-fired by means of oil burners, and uses preheated air obtained from cooling the sinter. The dust is collected by water sprays, thickened and pumped to the kiln feed. The overflow from the thickener tank is re-used for spray water, as this water contains sulphate derived from the sulphur in the fuel oil and cannot be added to the recirculating tailings water because it is very deleterious in flotation.

The kiln has a control room containing various automatic controls and graphic records of various thermocouples located in different parts of the kiln, as well as records of oil consumption.

TABLE 2.—*Analyses*

Size	Weight Per Cent	Cumulative Per Cent
+3 in.	4.77	4.77
+1.05 in.	23.86	28.63
+0.37 in.	44.50	73.13
+10 mesh	21.12	94.25
−10 mesh	5.75	100.00

Constituents, Per Cent

Mn	SiO_2	Fe	Al_2O_3	CaO
50.5–51.0	8.3–11.0	2.8–3.8	3.1–3.6	6.7–8.3

Final Products.—The final product from the kiln consists of hard, porous nodules light steel blue in color. Table 2 shows typical screen and chemical analyses.

AUXILIARIES

Suitable ores are jigged in a separate plant consisting of four sections of four cells each of wooden jigs. The launders are so arranged that both a 40 and 44 per cent concentrate can be produced. The concentrates are loaded directly into railroad cars; all plant rejects containing mineral that cannot be separated from the gangue by jigging are conveyed to the mill for flotation treatment and subsequent sinter production.

Sinter Machine.—A sintering plant containing a 72-in. by 88-ft. Dwight-Lloyd sintering machine is used as a standby for the kiln nodulizer. Installation of a 1500-ton concentrates-storage bin ahead of the kiln, plus the 350-ton mill storage bins, allows continuous operation of the mill regardless of kiln shutdowns for rebricking.

DISCUSSION

(*E. W. Engelmann presiding*)

C. E. LOCKE,* Cambridge, Mass.—The present successful operations in Cuba were the result of a long, hard battle. The paper does not give the reader any realization of the difficulties. One major problem was the occurrence of low-grade wad, which tended to form slimes that were difficult to treat and also caused the production of a lower grade of concentrate. Where the mineral was clean pyrolusite, the concentrating problem was easy but, unfortunately, the ore coming to the mill had in the past varied greatly in composition and in mineral character, which literally made the flotation operators tear their hair because flotation conditions were constantly changing and the operators could not keep up with them.

Earlier attempts to treat this ore were in the form of jigs, which would do good work on the pyrolusite. Later all-flotation procedure was

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adopted and this would have been reasonably successful if feed could be kept uniform in character and amount. The present combination of gravity concentration and flotation seems to be the most satisfactory process for this material, and the company should be congratulated on having stuck to the job and reached its objective.

C. R. Millar,* London, England.—This paper by Mr. Norcross tells a story of courage and efficiency, and calls for congratulations to everyone concerned, including the author of the paper.

The value of the paper would be much enhanced if the author could give us the following information:

1. The average percentages of Mn, SiO_2, Fe, Al_2O_3 and CaO of the ore treated over a period of six months or so.

2. Percentage of manganese recovered.

3. Quantities of reagents, fuel oil, and furnace linings used per ton of sinter produced.

4. Could the high silica content, 8.3 to 11 per cent, be substantially reduced? Is this amount of silica required to nodulize the sinter or is it unavoidably included in the concentrates?

5. Does removal of adhering sinter appreciably shorten life of furnace linings?

6. Power consumption in treatment plant, excluding furnace, per ton of sinter produced.

7. Disadvantages of D. W. machine as compared with Smidth furnace, relative economies and difference in operating costs.

F. S. Norcross (author's reply).—The following percentages answer Mr. Millar's first question:

Grade of Ore	Year 1943	Percentages				
		Mn	SiO_2	Fe	CaO	Al_2O_3
High......	Feb. 1–28	21.19	30.96	4.56	6.18	10.50
Medium...	Apr. 1–30	19.29	30.80	4.64	4.25	11.04
Low.......	May 1–31	16.56	35.00	4.10	4.85	12.00

2. Flotation recoveries of manganese have been: high-grade ore, 79.17 per cent; medium-grade, 79.79; low-grade, 78.85.

3. Quantities of reagents, fuel oil and furnace linings used per ton of sinter produced—not per ton of ore—are:

Ore	Fatty Acid, Lb.	Gas-oil, Gal.	Caustic, Lb.	Quebracho, Lb.	Lime, Lb.	Fuel Oil, Gal.
High-grade	19.11	12.39	5.61	1.50	7.55	1.32
Medium-grade....	24.53	16.32	8.29	2.89	9.58	1.58
Low-grade.	17.43	10.93	5.80	1.85	5.88	1.21

4. Under certain conditions, the silica has been reduced to 6.5 per cent, but there are two difficulties; one, that we have a lower extraction in the mill, and the other, that we have difficulty in sintering the ore because the silica content is too low to combine satisfactorily with the other bases, and although some sintering is perfectly feasible with this low silica the operation is considerably slowed up and the production of fines in the sinter product increases.

5. If adhering sinter is removed with care, and if the bricks have not been loosened by weaving of the shell, removal of sinter does not appreciably shorten life of furnace linings.

6. Power consumption in treatment plant, excluding furnace, per ton of sinter produced, is as follows: for high-grade ore, 82 kw-hr.; for medium-grade, 109, and for low-grade, 76.

7. Comparing the Dwight-Lloyd sintering machine with the Smidth rotary kiln on the basis of medium coarse feed with 15 per cent moisture showed that the capacity of the kiln was greater and that the dust losses were less.

The principal advantage to our operation of the Smidth kiln was the ability to feed very fine flotation concentrates in a sort of mash, or a slurry containing from 28 to 34 per cent moisture and with a very low dust loss. There were three big savings to us in the operation of the Smidth kiln in the order of importance: (*a*) the ability to feed fine concentrates and rich manganese slimes (which otherwise were lost), thereby increasing the extraction very greatly; (*b*) much lower dust losses; (*c*) lower operating cost. In other words, the kiln was particularly adapted to our process whereas the Dwight-Lloyd sintering machine was not.

* Mining Engineer.

Driving a 540-foot Raise at Nivloc, Nevada

By R. K. Matheson,* Junior Member A.I.M.E.

(New York Meeting, February 1942)

The Nivloc mine is 9 miles west of Silver Peak, Esmeralda County, Nevada. It has been operated by Desert Silver, Inc., since the summer of 1937. The cyanide mill treats 190 tons of silver-gold ore per day.

In March of this year the main operations were about 2000 ft. west of the main shaft, which served as the downcast air passage for the mine. The mechanical blowers in use were sufficient to ventilate the west end at that time, but it was foreseen that as the mining area was extended ventilation would be an increasingly difficult problem, especially during the summer months.

It was decided, therefore, to continue a raise from a point 100 ft. above the highest mining level (440-ft. level) to the surface; a distance of 443 ft. at approximately 65°. From this point 40 ft. of drifting would be necessary to connect with a shallow shaft on the surface. (Fig. 1.)

All the raise was serviced from the 440-ft. level; costs and progress obtained in the initial 100 ft. are included in the following pages.

Choice of Raise

A raise in this locality would have four objects: (1) to provide an ample supply of fresh air to the working places; (2) to explore the vein from the highest mining level to the surface; (3) to provide waste fill for stopes in the area, and possibly provide a transfer for waste from the surface; (4) to provide an emergency exit in the west end of the mine.

Before the mining of an ore shoot in the eastern part of the mine, a similar raise had been driven for purposes of ventilation and exploration. This raise and subsequent mining showed the vein to be noncommercial for a distance of about 250 ft. below the surface. However, in the area under operation, old surface workings showed occasional values. Thus from a development standpoint the raise was an excellent prospect.

The raise chosen was a three-compartment, stulled raise having two chutes and a manway. The manway inside timber measured 4 ft. 8 in. by 4 ft. 6 in.; it carried air and water pipes, a ladderway and a slide for a timber skip.

The two chutes would be used for drawing off the muck and the cleaning of the bulkhead, as well as for ore and waste passes, which would be especially useful for lateral exploration from the raise. The fairly large cross section of the raise (6 by 15 ft.) would provide a large quantity of waste for backfilling, and more could be obtained from a glory hole on the surface.

Time Schedule

As speed in completing the raise was the main requisite, three shifts were considered, but two shifts were decided upon when it was found that three would necessitate considerably larger fans than those available for clearing the raise. The main diffi-

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* Mine Foreman, Desert Silver, Inc., Nivloc, Nevada.

culty was not in exhausting powder smoke from the raise, but in preventing smoke from entering it.

The two-shift schedule provided an 8-hr. interval between blasting and the beginning of the timbering shift. The raise was cleared by blowing drill air and this was further facilitated by keeping the muck in the chutes well below the face of the raise.

The seven-hour collar-to-collar schedule in force at the mine made work difficult for the raise crews, as it meant that only about six hours were available, on an average, at the face. However, the men never missed a round during the whole operation.

The initial 100 ft. was driven by four men in 15 working days. The remaining 443 ft. of raise, 20 ft. of station and 40 ft. of drift required 5 men working 85 days.

The raise was driven on contract, the men sharing equally in the bonus. Raising and drifting were paid by the foot, while station-cutting was contracted by the cubic measure excavated. All supplies, including explosives, were furnished by the company.

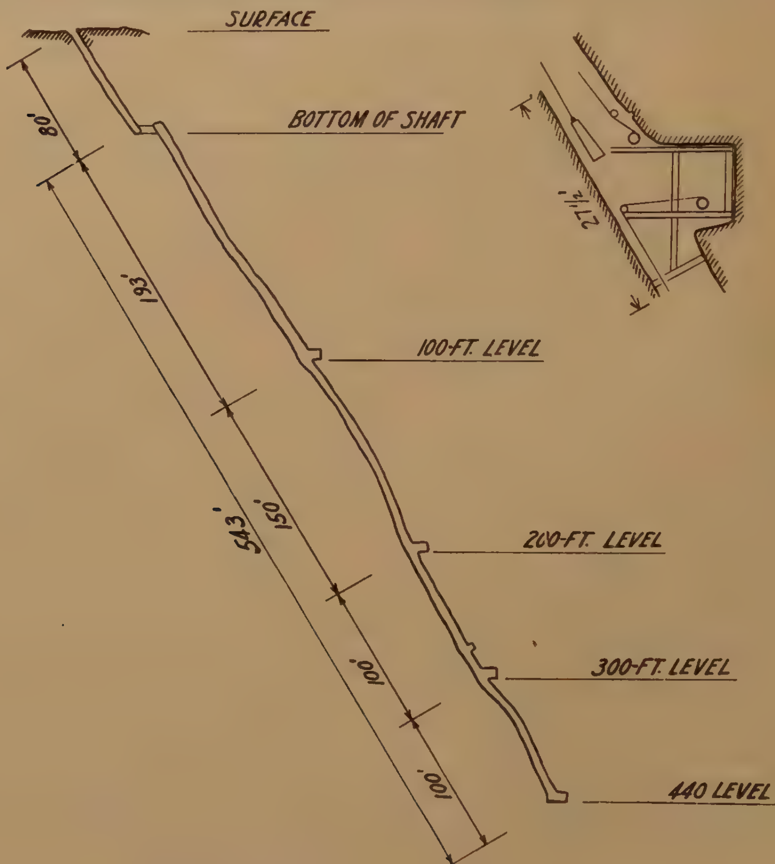

FIG. 1.—SECTION THROUGH 540-FOOT AIR RAISE.

MINING AND TIMBERING

The drilling and blasting were done by two miners working night shift. Ingersoll-Rand stopers and Timken detachable bits were used.

The ground varied from hard vein quartz to brecciated and gougy sandstone. A standard V-cut round, comprising from

eighteen to thirty-six 6-ft. holes as required, was used throughout. The powder per round varied from 150 to 200 sticks of 45 per cent strength.

cleaned the bulkhead. One man would then hoist the timber while the other two would block it in place.

The set comprised two 8 by 8-in. stulls

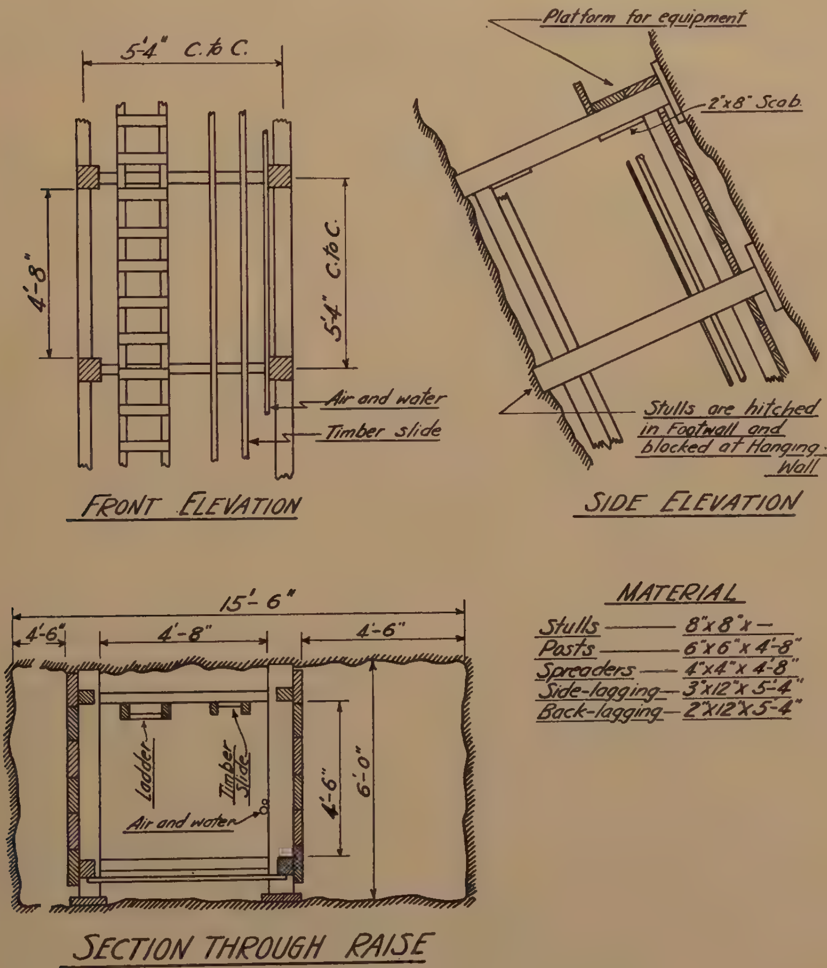

FIG. 2.—TIMBER DETAIL, 540-FOOT AIR RAISE.

A pipeman on company account carried a 2-in. air line and a ½-in. water line within 50 ft. of the face. Water pressure for the last 200 ft. was maintained with two 30-gal. pressure tanks.

The timbering was done by three men working day shift. Two of these men hoisted timber from the 440-ft. level while the third man scaled down the raise and placed with the ground on 5-ft. 4-in. centers laterally and vertically (Fig. 2). The stulls were braced with 6 by 6-in. posts and girts on both walls, and were lined with 3 by 12-in. lagging. Lining was also carried on the hanging wall over most of the distance. Three thicknesses of lagging were used on the bulkhead, which was carried within one foot of the back at times. On an average,

the crew gained one set about every six days, and on the sixth day the timbermen placed two sets. They also usually set the stagings for the drillers.

Fig. 1. A spare set of timber, water tanks, spare drills and other material were also stored in the stations.

The raise was serviced with two 4-hp. Ingersoll-Rand air hoists, which were carried up the raise and placed at successive stations as the raise advanced. Two 5-ft. by 18-in. timber skips, running on 2 by 4-in. guides were used.

The contractors handled the muck through the first grizzlies; below them it was handled by men on company account as required.

TABLE 1.—*Progress of Work*

Work	Drilling Shifts	Timbering Shifts	Total Shifts	Advance, Ft.	Advance per Shift, Ft.
Raising	86	88	174	543.0	3.12
Station cutting	9	10	19	20.0	1.05
Drifting	7		7	40.0	5.71
Totals	102	98	200	603.0	

TABLE 2.—*Costs*

Work	Drilling		Timbering		Explosives	Labor	
	Labor	Supplies[a]	Labor	Supplies		Total	Cost per Ft.
Raising	$2,099.57	$807.00	$3,420.07	$2,127.12	$620.45	$5,519.64	$10.15
Station cutting	220.20	84.40	389.70	243.10	65.97	609.90	30.45
Drifting	170.90	65.70			50.60	170.90	4.27
Totals	$2,490.67	957.10	$3,809.77	$2,370.22	$737.02	$6,300.44	

Work	Supplies		Direct Cost	
	Total	Cost per Ft.	Tota	Per Foot
Raising	$3,554.57	$ 6.54	$ 9,074.21	$16.69
Station cutting	393.47	19.63	1,003.37	50.08
Drifting	116.30	2.91	287.20	7.18
Totals	$4,064.34		10,364.78	$17.18

[a] Includes proportional charges for air, blacksmithing, and miscellaneous.

HANDLING SUPPLIES

Stations were cut into the hanging wall at distances of 100 ft., 200 ft. and 350 ft. above the bottom. These stations were about 6 ft. wide and extended the length of the raise. They were timbered with framed 8 by 8-in. timber and lagged. Two chutes were placed at each station, as well as two 10-in. grizzlies.

All equipment and supplies were handled from these stations. The two hoists were usually set here, arranged as shown in

PROGRESS AND COSTS

Progress and costs are shown in Tables 1 and 2.

Development and Dewatering Practice at Park City Consolidated Mines

By Gloyd M. Wiles,* Member A.I.M.E.

(Salt Lake City Meeting, September 1940)

The eastern section of the Park City district is drained to an elevation of 6300 ft. by means of the Ontario drain tunnel owned and maintained by Park Utah Consolidated Mines Co. This elevation represents a depth of 920 ft. in Park City Consolidated Mines Company's property.

When development below the 900-ft. level was commenced, sinking was attempted through an incline winze in the fissure. The intensely fractured quartzite forming the walls of the vein could not be held after water was encountered, so it was decided that further sinking should be done in a vertical winze far enough in the footwall to be in the more solid quartzite outside the fractured zone containing the vein. Progress was good until a depth of 200 ft. below water table was reached, where the flow of water became serious. Sinking progressed 300 ft., to the 1200-ft. level, and there approximately 750 gal. of water per minute was being pumped. The last 20 ft. of sinking cost more than $200 per foot direct charges, whereas the over-all cost for the 300 ft. was $100 per foot. This high last cost was caused by excessive lost time in pumping.

Crosscutting to the vein disclosed the fact that the 1050-ft. level had been developed only after drainage had been effected from the 1200-ft. level, but that the 1200-ft. level could not be opened because of the great amount of water encountered when the fractured zone was intersected. The ground in the fractured zone is so finely crushed that it precludes any possibility of economic support when it is wet. Furthermore, mining costs were found to be prohibitive after caving of vein walls had taken place as a result of an attempt to drive drifts before complete drainage had been effected.

More than 3000 gal. per minute was pumped from the 1200-ft. level and the water table remained 75 ft. above this plane even after five crosscuts were driven to the fractured zone at 50-ft. intervals along the strike of the fissure from a footwall drainage drift.

It was evident, therefore, that complete drainage must be effected before the fractured zone could be opened in any future development. After extensive investigations in this connection, a well-drilling campaign was decided upon to effect drainage of the fissure in advance of sinking and level development, so that all mining operations would be carried on in dry ground.

Drilling Procedure

A crosscut was driven through the hanging wall of the fissure on the 1050-ft. level, which was the deepest dry level at the time, to a predetermined location from which a vertical hole should reach the fractured zone at about 400 ft. below this level. A drill station was made and a raise was driven over the location for the hole, to act in lieu of a derrick or drill tower. This raise

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* Vice President and General Manager, Park City Consolidated Mines Co., Park City, Utah.

was driven through to the 900-ft. level and room was made at its top for a power-switching station for pump feeders and controls (Fig. 1).

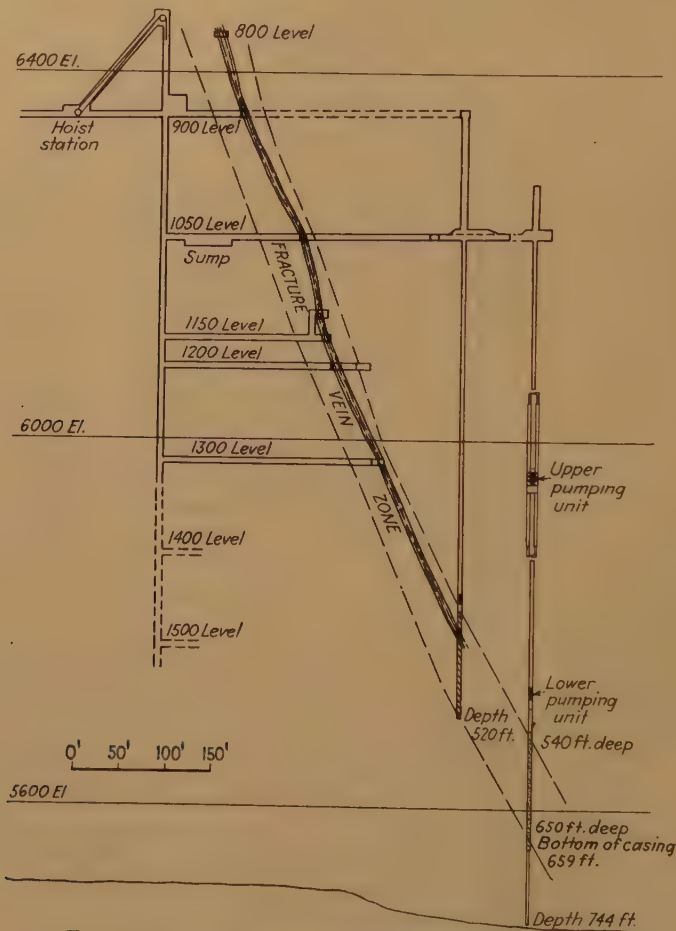

FIG. 1.—VERTICAL SECTION SHOWING DRAINAGE DIAGRAM.

for 520 ft., where the hole bottomed in solid quartzite footwall rock. The fractured zone in this hole extended from 400 to 500 ft., and although considerable caving occurred

Roscoe Moss Co., of Los Angeles, constructed a drill rig for this work and supervised subsequent installation and drilling. While the solid quartzite to be drilled was extremely abrasive and required the use of an expert welder to do the bit dressing, it permitted the use of untrained helpers, so that an actual saving in labor cost was effected. The hole was started at about 23-in. diameter and maintained at this size in this section, no appreciable delay in drilling time resulted. Bit wear and consequent dressing caused most of the delay. Although numerous welding-rod steels were tried, the most satisfactory results were obtained by building back to gauge with Chicago Steel and Wire welding rod Nos. 110, 350 and 550 successively, and then laying a one-bead surface of type T (tungsten carbide) for the final dressing.

The well was cased with lapweld pipe of 20-in. outside diameter and ⅜-in. wall thickness, with welded joints. The section to be installed through the fractured or water-bearing zone was perforated with vertical slots, each ⅛ by 6 inches.

A type 5-K Byron Jackson five-stage turbine pump, having a capacity of 2250 gal. per min. at 500-ft. head, was installed in the well. It was suspended from 10-in. spiral welded, 10-gauge Victaulic-coupled pipe and was direct connected to a 350-hp., 2300-volt Byron Jackson submersible motor. After about 1100 gal. per min. was produced, the pumping unit was removed. A scow was used to clear the perforations and to clean out the well.

The pump was reinstalled but only slight improvement was noticed in the flow. The pumping unit was again removed and a special perforator, designed to slot casings already in place in the well, was used to add additional perforations ½ in. wide by 6 in. long. This improved the flow to 1400 gal. per min. A 200-hp. unit, more suitably fitting the conditions than the 350-hp. unit previously used, was installed, and continuous pumping from this well began to lower the water table almost immediately.

With the hope of developing a larger flow of water, a location was selected on the 1050-ft. level for No. 2 well about 225 ft. to the southwest along the strike of the vein and sufficiently far in the hanging wall to reach the fractured zone 125 ft. deeper than No. 1 well.

In drilling No. 2 well, a soft, narrow limestone bed was found at a depth of 80 ft. and when hard quartzite was reached below it the dip of the beds caused a crooked hole. Since this was above water, a man with a Jackhamer was lowered on a rope to cut a hitch in the bedding plane, so that the hole was made straight within a few minutes. A similar bedding caused the same difficulty some distance below the water table and approximately three weeks was required to get the hole straight again. Eventually this was accomplished by filling the crooked part of the hole with hard boulders, cast-iron scraps, and old wire rope, again and again, until the bit wore away the bedding angle of the rock and cut a hitch with a horizontal bottom, permitting straight-hole drilling once more.

This hole reached the fractured zone at 540 ft., approximately as planned, and was bottomed in footwall quartzite at 659 ft. A 12-in. diameter prospect hole was drilled from 659 to 744 ft. total depth. This latter section was not cased. The upper 331 ft. were cased with lapweld pipe of 20-in. outside diameter, similar to that used in No. 1 well, and No. 6 gauge, double well casing of 18-in. inside diameter, having stovepipe joints, was installed from 331 to 659 ft. The stovepipe casing situated in the fractured zone was perforated with horizontal slots, ⅜ by 1¼-in., by punching the sheets prior to fabrication of the pipe.

Motors and Pumps in Series

The pumping unit originally used in No. 1 well was installed in No. 2 well to a depth of 525 ft. and, although 3000 gal. per min. was pumped, the water was lowered only to 264 ft. below the collar of the well. Since this amount of draw-down would not accomplish sufficient drainage, a design for two submersible motors and pumps in series was worked out, whereby the combined capacity would be approximately 4500 gal. per min. against 500 ft. of head.

The 10-in. discharge column used in the first instance was too small to handle so great a quantity of water without excessive head loss, so it was necessary to use a 12-in. column. The 12-in. Victaulic couplings were larger than the well casing, so threaded and coupled standard pipe was used. (Three hours time was required to install the unit with Victaulic-coupled discharge pipe, but almost 20 hr. was needed when threaded column was used.)

The two motors are interlocked electrically, so that failure of one will stop the other. In operation, the lower pump starts first and a time delay relay starts the upper pump 15 sec. later. A pressure switch near the top of the column shuts down both pumps if the water fails to arrive or if the well should run dry at any time. The starting switches are actuated by a hand-off-automatic push-button switch controlled by a float switch in the gathering sump on the 1050-ft. level where the wells discharge the water for transfer to the 900-ft. level. A battery of six centrifugal station pumps at 1200 gal. per min. and one close-coupled deep well turbine pump at 4000 gal. per min. accomplishes this last lift to the 900-ft. level. This float switch shuts down the well before the gathering sump can overflow and starts the motors again at a predetermined lower level. Electrical devices at the discharges from both wells operate an alarm system to the surface, indicating failure if water is no longer discharged from either or both wells.

Even though one motor cannot operate more than a few seconds after failure of the other, by reason of the electrical arrangements described above, the Byron Jackson company did not wish to chance making a continuous discharge column through the two units to the surface, for fear the lower pump might fail and permit the upper unit to pull a vacuum over the mercury seal. It has not been determined definitely whether pulling of a vacuum might not cause the oil to flow out of the motor through the hydrostatic balance tube (Fig. 2). It was necessary, therefore, to construct a tank to contain the upper unit, which would remain open at the top to prevent any possibility of the vacuum being formed. A 16-in. outer column, 75 ft. long, was extended above the upper unit to allow for equalizing of the hydraulic loads between the two motors (Fig. 1).

As a result of the complicated mechanical design and the high cost of removal of this equipment from the well for repairs, ratchets are inserted in the couplings between the pump and the motor whereby impeller adjustment can be effected as wear progresses, merely by changing the leads to the motors and reversing them. Each time the motor is reversed, the impellers in the pump are dropped 0.003 in. Repeated reversals, therefore, will result in setting the impellers down against the bowl liners and in keeping the pumps up to full efficiency until the impellers or liners are worn to the place where replacement of these parts becomes essential.

This equipment has been in operation 90 days, is now producing 4300 gal. per min. against 480 ft. of head (all the well will furnish) and has lowered the water table 175 ft. Subsequent sinking from the 1200-ft. level to the 1300-ft. level while handling approximately 75 gal. of water per minute was accomplished with direct charges amounting to $70 per foot, compared with the cost of the last 20 ft. approaching the 1200-ft. level, where 750 gal. per min. was pumped at a cost of more than $200 per foot. It will be remembered, also, that the 1200-ft. level could not be opened under the previous dewatering scheme, as it was impossible to support the ground in the fractured zone when it was wet. With the water previously drained from the 1300-ft. level, however, no unusual ground-maintenance problems were presented in prosecuting development through the fractured zone or in the vein. The method of dewatering presently being used has effected material savings and has proved entirely satisfactory from every standpoint.

Submersible Motors

The motor is mounted below the pump. A cast-iron grid type of strainer connects the motor to the pump, and the entire assembly is suspended from the surface by the column pipe. Water developed from strata below the motor flows up past it and

enters the strainer, passes through the pump into the discharge column to the 1050-ft. level and is discharged into an open sump.

closing the entire motor in a case filled with oil of high dielectric strength. Prevention of oil leakage from the motor at the point where the shaft passes through the case,

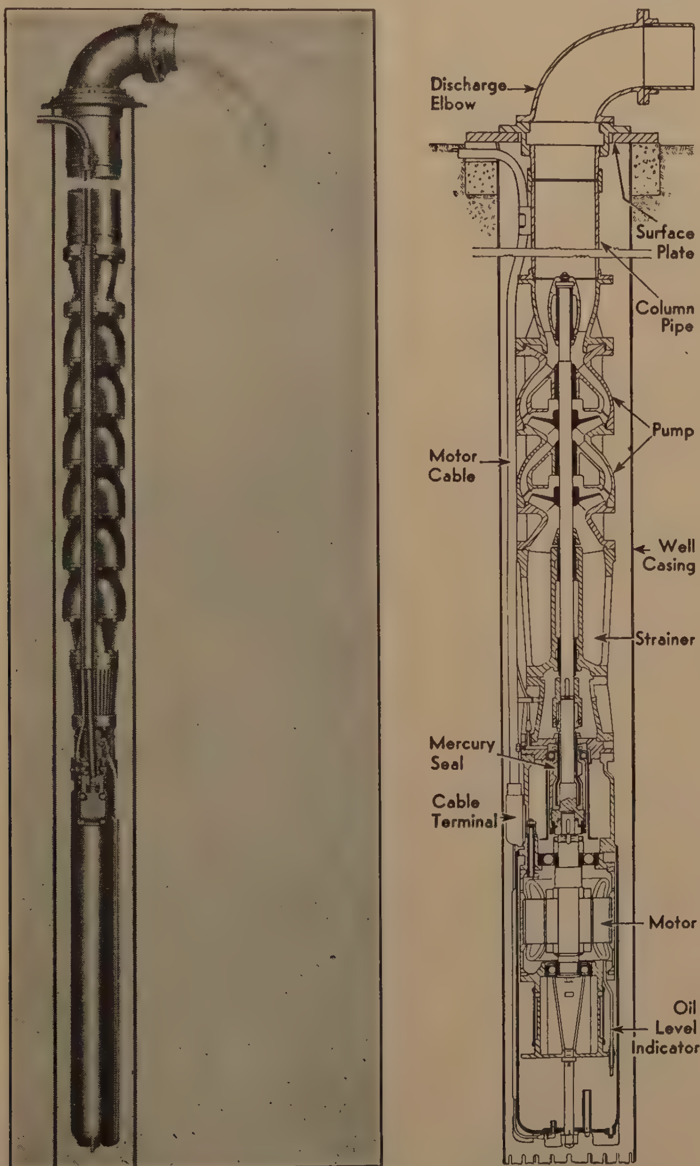

FIG. 2.—DETAILS OF PUMP ARRANGEMENT.

Moisture penetration into the active parts of the motor was eliminated by en- and also of water entering at the same point, was accomplished by the use of a

mercury seal. The drive-shaft extension to the pump is from the top of the motor. Mounted on the shaft, and rotating with it, is a cylindrical cup partly filled with mercury. Dipping into this cup is a stationary, cylindrical, hollow baffle, through which the drive shaft extends. The cup, or seal chamber, and the baffle, are designed with hydraulic lines that permit minimum displacement of the mercury during rotation. Erosion and corrosion of the chamber walls, which would occur with ordinary steels, are avoided by the use of high-chrome-nickel steel in the chamber, and an enameled baffle.

The mercury seal at the upper end of the motor, through which the rotor shaft extends, constitutes the barrier between oil in the motor and water in the well. Substantially, the seal is a rotating U-tube. On one side, oil contacts the surface of the mercury lying between the inner wall of the seal chamber and the outer wall of the baffle. On the other side, water from the well is free to contact the mercury in the area between the inner wall of the baffle and the outer surface of the rotating shaft. Any contact or possible mingling of the oil and water is prevented by the mercury in the seal chamber.

To balance the hydrostatic forces on the two sides of the seal, and to compensate for the heat expansion of the liquid in the motor, it is necessary to introduce a balance tube, which connects the water and the oil side of the seal. The balance line leads from the chamber on the water side of the seal, which is under hydrostatic well pressure, to a balancing chamber in the bottom of the motor—the bottom portion of this chamber being filled with water. This balancing chamber is in direct pressure communication with the motor chamber, through a small opening around a metal diaphragm of practically the same diameter as the motor case. The result is a quiescent chamber, which is at the same pressure as that due to the submergence in the well, with the pressure on the oil side of the mercury seal always the same as on the water side. This balancing device, together with the cable and oil-filler pipe, forms a U-tube, with the oil from the motor rising in the cable and oil-filler pipe, to a height sufficient to balance the well pressure.

During operation, the oil within the motor case is warmed, and as it expands forces some of the water from the bottom of the balance chamber through the balance tube, and into the well. Were direct communication between the motor chamber and the well not possible, internal pressures would be built up that would force mercury to spill out of the seal. Ample cooling is secured by providing passages to permit continuous oil circulation from the windings to the shell.

Power is supplied through a lead-sheathed steel-armored cable extending from the collar of the well to the terminal box in the upper end of the motor.

Driving a Tunnel in Fractured Rock Formation Carrying Water under High Static Pressure

By S. H. Ash,* Member A.I.M.E. and P. S. Miller†

(New York Meeting, February 1942)

Extensive and diversified resources justify large populations and great industries. To carry on the business of commerce and meet the demands of large populations, the utilization of tunnels in some form for transportation purposes constitutes an important link in modern civilization.

The driving of tunnels has been appropriated by the tunneling branch of the heavy-construction industry, and the progressive steps taken by modern tunnel builders were not even dreamed of 50 years ago. Great strides have been made toward more rapid progress in tunneling in the United States during the past 10 years.

Tunneling consists of a definite cycle of operations developed from ideas that are crystallized and translated into action programs through various engineering phases that begin with the first preliminary survey and do not end until the job is completed. Once the line of the bore is established and a time set for the completion of the work, the face must advance according to estimates prepared before the work commences. This schedule must be followed in regular sequence from the moment the drills start against a new face of rock until they are ready again to drill the succeeding face, constituting a round.[1]

Mining is largely tunneling carried on under conditions often more inherently hazardous than actual tunnel construction, but for various reasons[1] tunneling is more hazardous than mining in any branch of the mineral industry. On the other hand, obstacles are sometimes encountered in heavy-construction tunneling projects that when met in mining enterprises do not offer comparable tasks to avoid or overcome them. One is the existence of water under high pressures. In mining enterprises, various alternatives offer a solution in driving a tunnel, even to the extent of its discontinuance or the abandonment of the mine, but a "construction" tunnel usually must go forward because some important project depends largely if not entirely upon it.

This paper describes an unusual method of driving a water-supply tunnel through rock carrying water under pressures and conditions that did not permit the use of ordinary tunneling methods.

The Delaware Aqueduct

The new Delaware aqueduct is a Manhattan project of some magnitude; when completed, it will contain what probably will be, for some time to come, the longest tunnel in the world. Various features in its construction are currently described in the trade journals of the construction industry, and informative engineering details are given in the *Delaware Water Supply News*, published twice a month by the Board of Water Supply of the City of New York. Work was begun on this project on March 24, 1937. The tunnel will convey water from reservoirs at the headwaters of the Delaware River to the City of New York.

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[1] References are at the end of the paper.

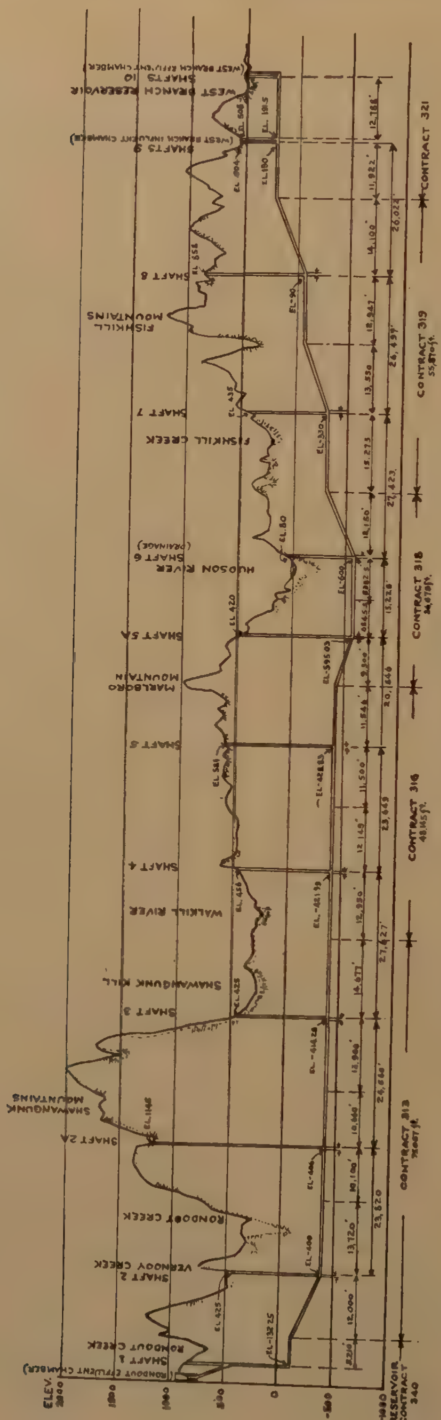


FIG 1.—PROFILE OF RONDOUT-WEST BRANCH TUNNEL SECTION OF DELAWARE AQUEDUCT.

The 85-mile Delaware aqueduct is being driven from 31 concrete-lined shafts 310 to 1550 ft. deep, a total depth of 19,307 ft. With its southern extension, City Tunnel No. 2, recently completed, it will make a continuous deep rock tunnel 105 miles long, and from 13 ft. 6 in. to 21 ft. in diameter inside the concrete lining. The shafts and tunnel will be concrete-lined throughout. On Sept. 20, 1941, the percentage of tunnel work completed was: Excavation, 98.9 per cent; concrete lining, invert 69.8, and side walls and arch 62.4 per cent.

Rondout-West Branch Tunnel

The Rondout-West Branch tunnel of the Delaware aqueduct is 45 miles long and comprises the section of tunnel between shaft 9 and the Rondout reservoir (Fig. 1).

In driving the Rondout-West Branch tunnel through the section known as Contract No. 313 (Samuel R. Rosoff, Ltd., contractor), certain hazards not commonly encountered in tunneling have been handled successfully. This particular section is 75,057 ft. long and is opened by three shafts (2, *2a*, and 3) having depths of 825 ft., 1551 and 839 ft., respectively. Throughout most of its length, the ground gives off methane and necessitated ventilation methods and practices similar to those used in coal mines and other gassy tunnels.[2,3] The tunnel excavation in Contract No. 313 was completed on Sept. 17, 1941, without the occurrence of a single gas explosion, although in several instances methane gas prevented progress in a heading. The lining of the bore with concrete was in progress in November 1941.

The cover above the tunnel under the Shawangunk Mountains between shafts 2 and 3 reaches 2400 feet.

In driving the section of tunnel between shafts 2 and *2a* (Fig. 1), it was necessary to pass beneath the Rondout Valley at Wawarsing; and from previous experience[4] it was anticipated that unusual water-bearing ground, consisting of glacial drift and a series of variable, steeply inclined and faulted beds of shale, sandstone and limestone, would be encountered for about 500 feet.

At the tunnel crossing, Rondout Creek lies at the southerly side of a wedge-shaped gorge approximately 3000 ft. wide and filled with thoroughly inundated glacial drift to a depth of more than 400 ft. between the bedrock and the present creek bed, which is about 650 ft. above the tunnel. This leaves a minimum depth of water-bearing rock cover of about 250 ft. over the tunnel at the deepest point of the gorge.

Nothing unusual in the nature of water was found in driving the tunnel 6778 ft. southward from shaft 2. Beyond this point, it was anticipated that precautions would be necessary to ensure safe and successful driving of the bore.

For this reason, test holes were drilled in advance of each round beyond the point mentioned above. These holes indicated the necessity for grouting, and the heading was advanced 185 ft. (6963 ft. from the shaft bottom), at which point it was decided that advance could not be made by ordinary grouting methods, and no advance was made in the south heading of shaft 2 from July 30, 1940, to June 15, 1941.

Freund[4] has described the grouting, drainage facilities, and drilling features in connection with exploration of the ground ahead of the face by core borings and the exploration and unsuccessful advance of a side drift (see Figs. 2, 3, 4, and 5). The exploratory work consisted of 3189 lin. ft. of core borings and, in addition, 1392 lin. ft. of holes, which were grouted to consolidate the ground further in the vicinity of the bulkhead and the blanket placed at the face before unsuccessful driving of the drift (drift *A*, Figs. 3, 4 and 5). This drift had been advanced about 87 ft. when the volume of water indicated that it was unsafe to proceed (Fig. 5). The drift was abandoned and refilled solidly with concrete.

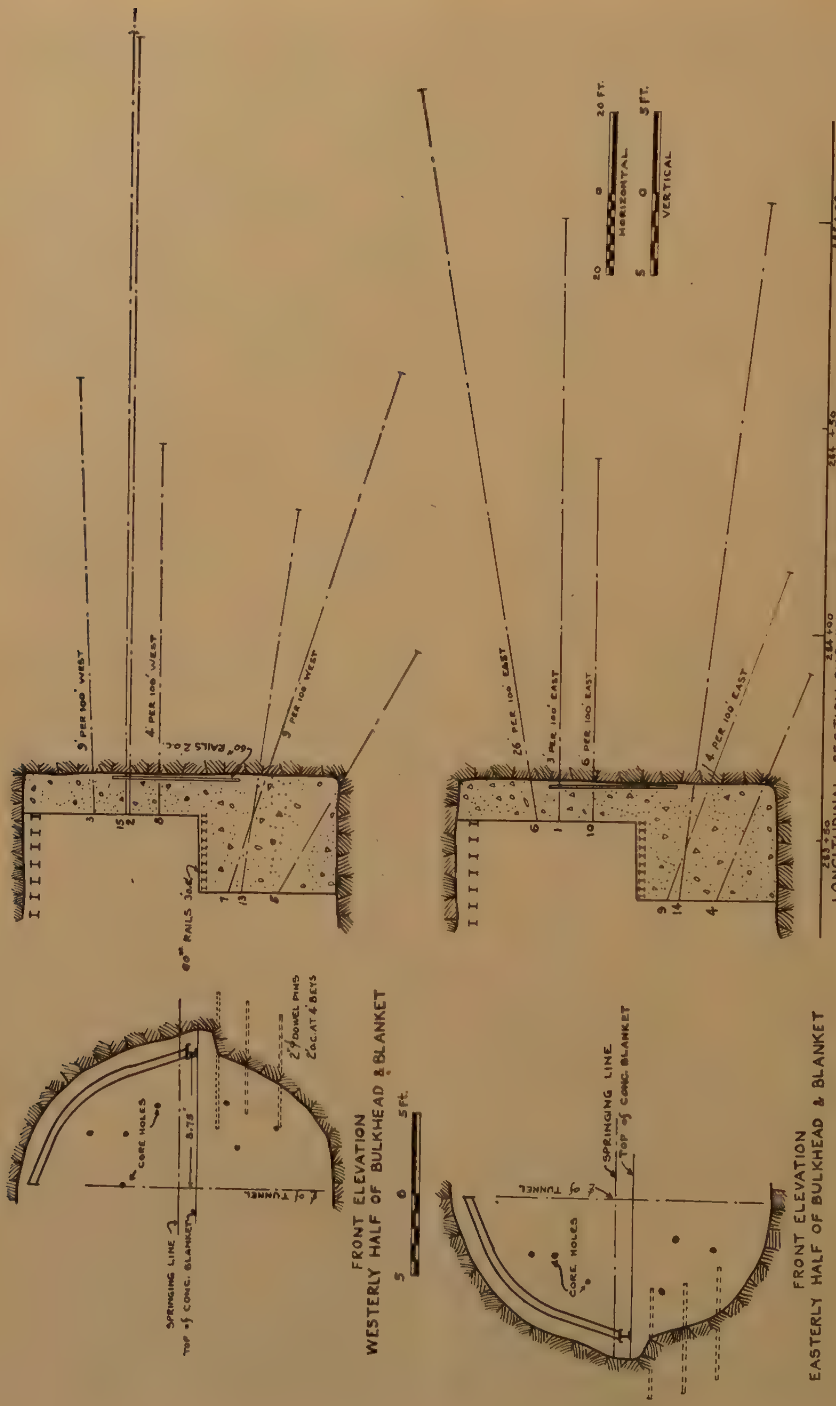


FIG. 2.—FRONT AND SIDE ELEVATIONS SHOWING DETAILS OF CORE BORINGS IN VICINITY OF BLANKET AND HEADING BULKHEAD.

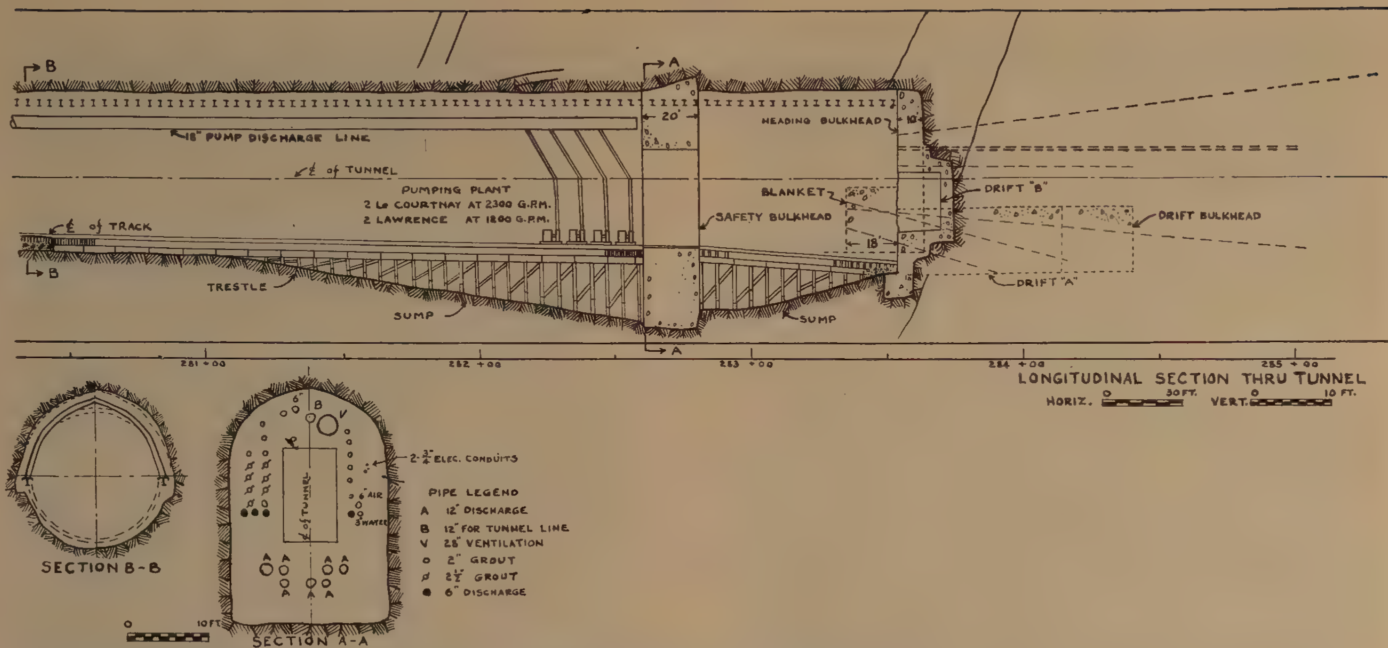


FIG. 3.—LONGITUDINAL SECTION THROUGH TUNNEL SHOWING LOCATION AND DETAILS OF DRIFTS, BLANKET AND HEADING BULKHEAD, SAFETY BULKHEAD, SUMPS, AND PUMP STATION.

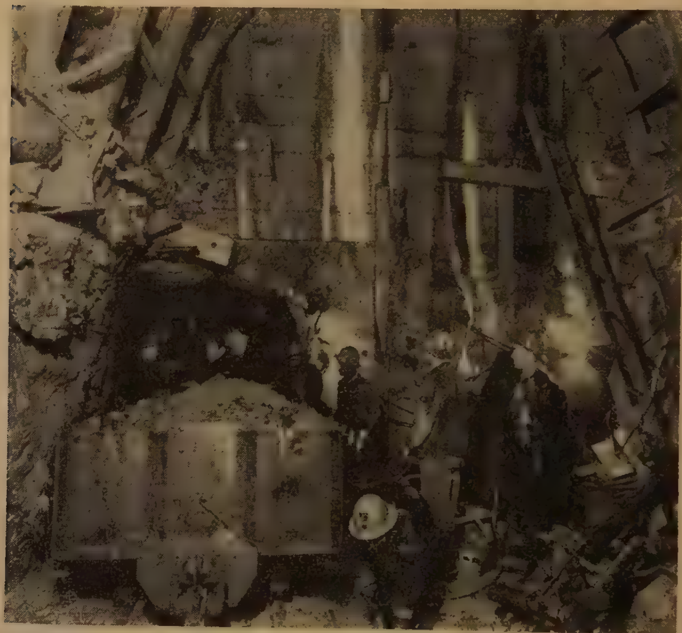


FIG. 4.—LOOKING SOUTH THROUGH PORTAL OF DRIFT A, LATER ABANDONED.
Concrete blanket has been removed in front of heading bulkhead. Observe all workmen wear head protection, and water is carried in ditch.

FIG. 5.—WATER POURING OUT OF DRILL HOLES AT FACE OF DRIFT A. THIS DRIFT WAS ABANDONED AND FILLED WITH CONCRETE.

Safety Features in Connection with Advance by Drifts

The exploratory work by core drillings indicated an average inflow of water (before grouting) of about 1000 gal. per min. and in some instances as high as 1500 gal. per min. The indicated inflow in drift *A* remained 2100 gal. per min. without diminution for 6 weeks, even after a 26-ft. concrete bulkhead had been placed in this drift (Fig. 5). The critical feature, however, was a consistent static pressure of 290 lb. per sq. in. on the water inflows in all instances, indicating the source to be definitely the level of Rondout Creek. Furthermore, there was no diminution of inflow from test holes, either when allowed to flow freely or when adjacent holes were drilled; the net result in such instances was merely an increase in the total volume of water to be pumped to the surface.

Heading Bulkhead.—As stated above, before the tunnel face was advanced by drifts through doubtful ground (thereby possibly releasing uncontrolled flows and flooding the tunnel), a bulkhead and blanket were placed at the face, and the surrounding rock was consolidated by grouting, 1838 bags of cement being used. Other safety features provided also will be described.

The heading face bulkhead was 10 ft. thick and constructed against the entire face. The integral blanket extension was carried back 18 ft. farther, with its top surface 1 ft. below the tunnel axis. The bulkhead was reinforced with vertical 60-lb. railroad rails spaced on 2-ft. centers, and the blanket was reinforced with similar rails on 3-ft. centers horizontally transverse to the tunnel axis and keyed to the rock sides by 2-in. dowels.

Safety Bulkhead.—A safety bulkhead (Figs. 3, 7 and 8) was constructed to protect the workmen and tunnel from an uncontrollable situation that might result from an inrush of water or failure of pumping facilities. This condition actually occurred on one occasion because of a breakdown of pumping facilities. Besides demonstrating that the safety bulkhead could withstand the anticipated water pressure, this circumstance revealed some efficiently sealed points of leakage.

The safety bulkhead was constructed of concrete 20 ft. thick, was well keyed into good rock, and was provided with a hinged steel door. It was placed to allow 73 ft. between adjacent faces of the safety and heading bulkheads. Necessary pipes for drainage, power cables, and other facilities were carried through the bulkhead. These pipes were provided with valves or plugs, depending upon their service. The 28-in. metal pipe used for ventilating the tunnel was also carried through the concrete and so constructed that the section adjacent to the safety bulkhead on the working-chamber side could be removed quickly. A flap valve, which could be operated automatically or manually, at the ventilating-pipe connection provided an effective seal against the water. The section of haulage track passing through the bulkhead was made easily removable.

In constructing the safety bulkhead, due consideration was given to the water pressure to which the bulkhead might be subjected, and its design was based upon a high factor of safety rather than upon calculation of the proper thickness of the bulkhead.[5] The design of the safety bulkhead contemplated a mass of concrete that would act as a cork, as is evidenced by its construction (Figs. 3, 7 and 8). The bulkhead is not only keyed into the rock but is tapered in the form of a cork so that any pressure against it caused by a static head of water would result in greater reaction against the ends of the bulkhead. Steel was unnecessary in the concrete, as it was much too thick to act as a diaphragm. Rather, an arching action probably would occur.

The steel safety door (Fig. 8) was constructed in two parts like a Dutch door, so that if it should be necessary to dewater the

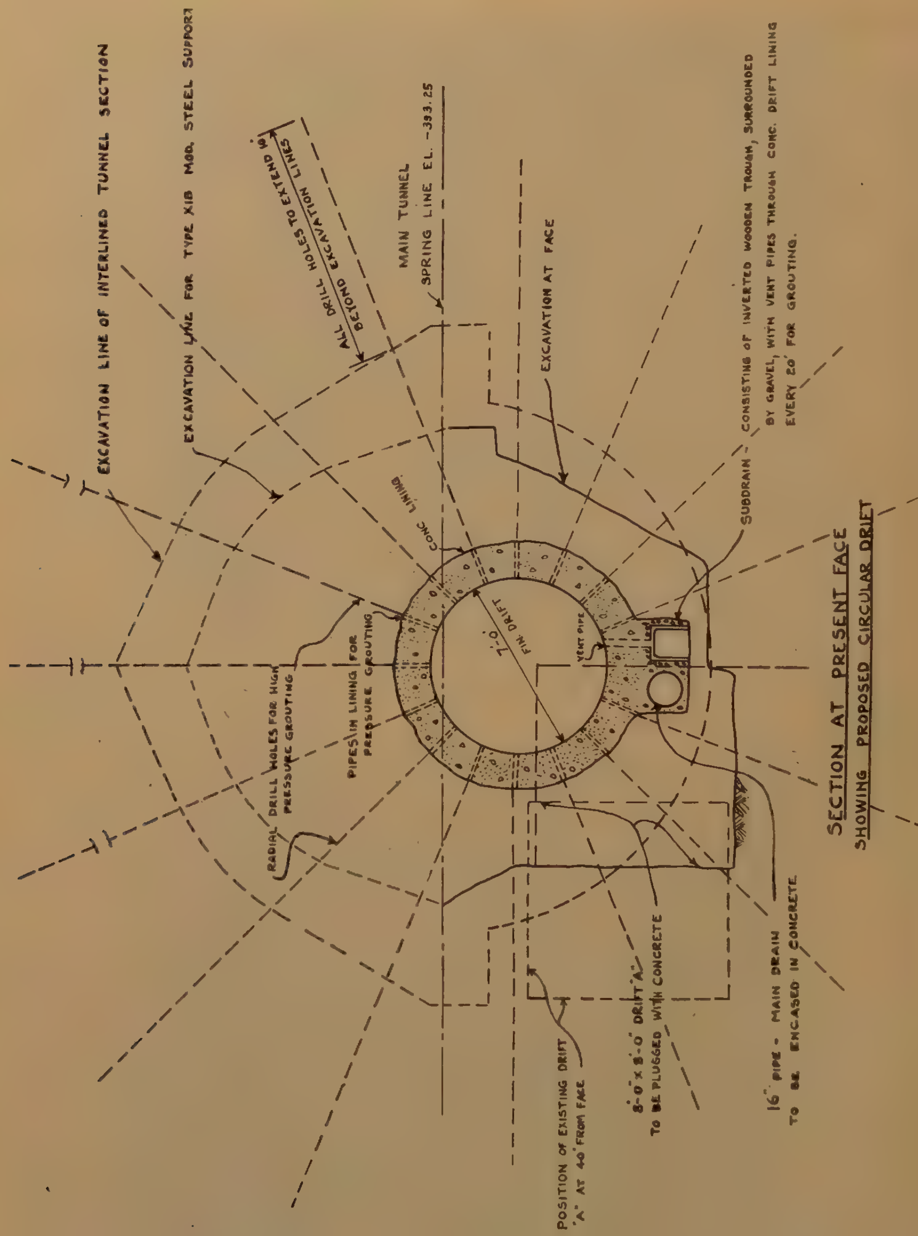


FIG. 6.—DETAILS OF CONCRETE-LINED PIONEER DRIFT *B*.
Shows arrangements for radial grouting holes used for consolidating mass of rock surrounding bore.

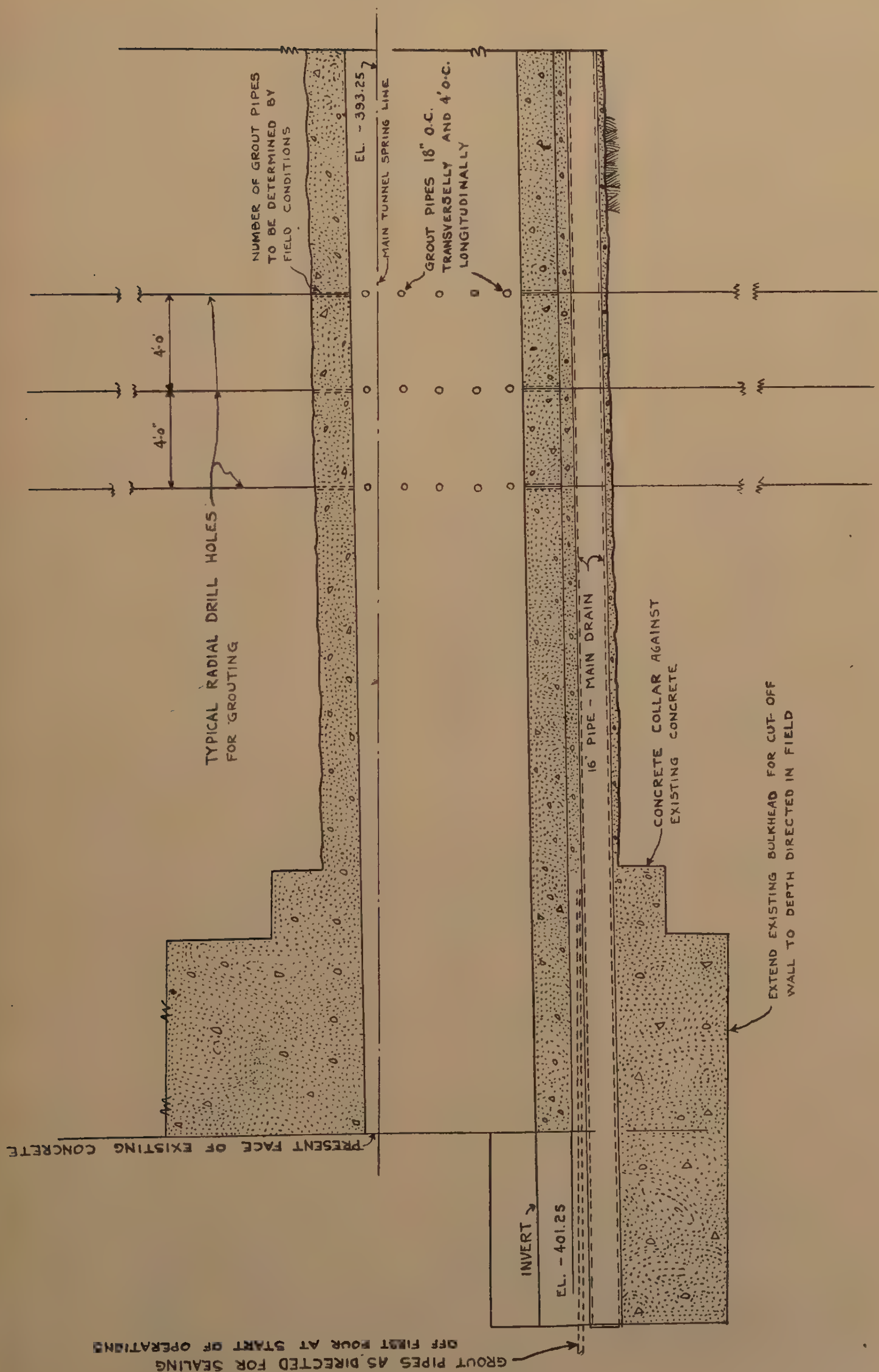


FIG. 6.—(*Continued*)

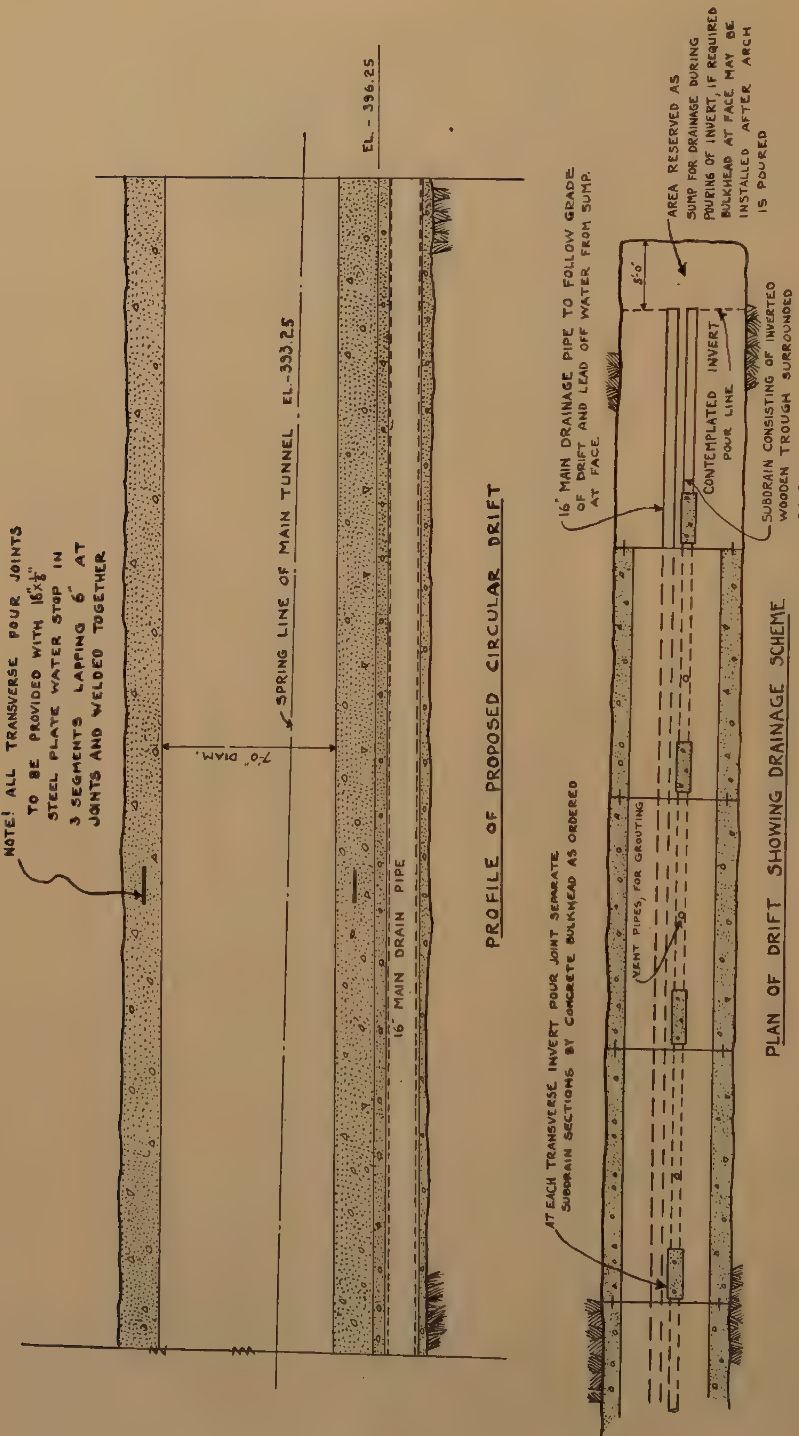
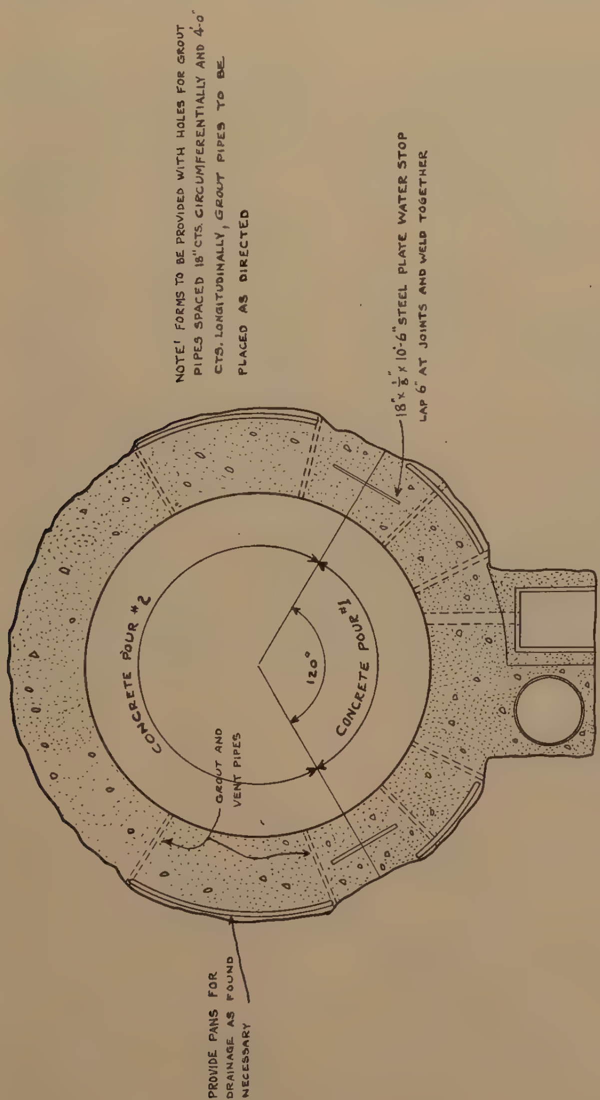


FIG. 6.—*(Continued)*

tunnel inbye the door the upper half could be opened when the water reached that level. A rubber gasket was used to permit a proper seal between the door and the steel doorframe embedded in the bulkhead.

The safety door (12 ft. 2 in. high by 7 ft. 6 in. wide) was made of 1-in. steel plate and stiffened in the back by horizontally placed 24-in., WF 94-lb. beams spaced 14 in. on centers. A pressure of 300 lb. per sq. in. was assumed.



SECTION OF DRIFT LINING SHOWING POURING SEQUENCE

FIG. 6.—(*Continued*)

Drainage Facilities.—Normal pumping facilities in shaft 2 consisted of units with an aggregate capacity of 1500 gal. per min., discharging for a lift of 825 ft. through two 6-in. discharge lines incorporated in the shaft lining. To provide for additional water from the water-bearing zone, pumping units with a capacity of 10,000 gal. per min. were installed at the shaft bottom, and three additional discharge lines, of 6-in., 8-in. and 12-in. diameter, were placed in

the shaft. For drainage from the heading to the shaft, it was necessary to provide an 18-in. spiral welded discharge line from the pipes placed in the lower portion of the safety bulkhead and that on the shaft side to be used as a pump sump, having a

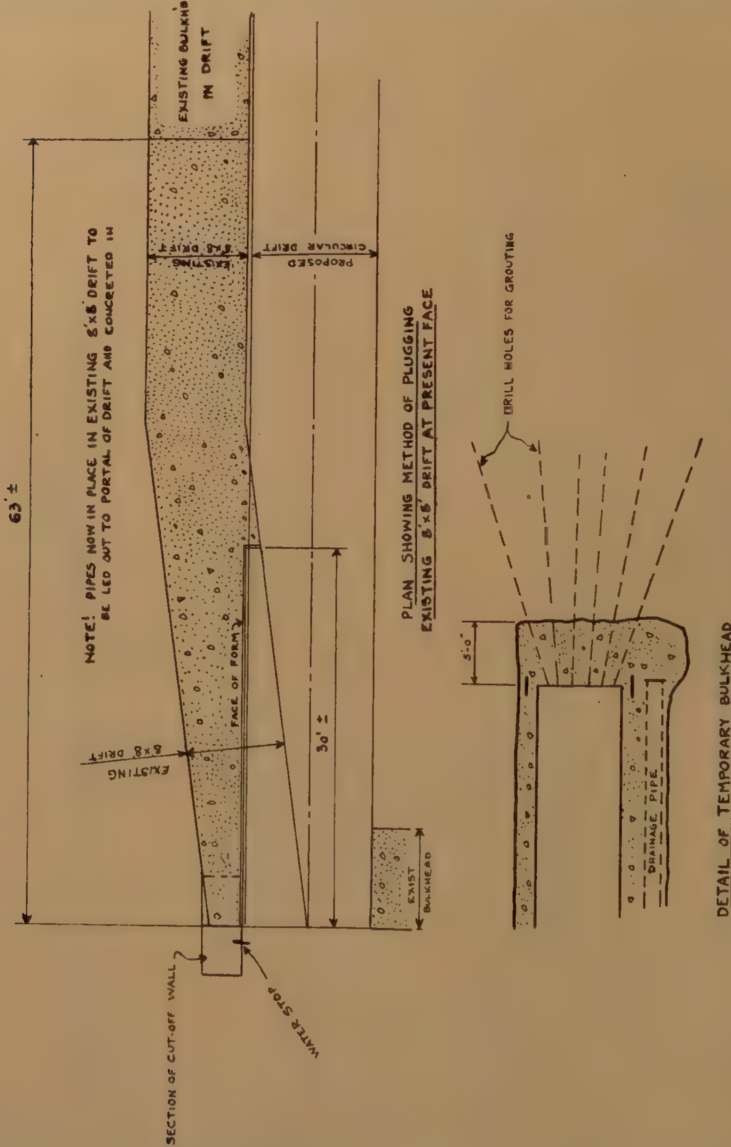


FIG. 6.—(*Continued*)

pump station at the safety bulkhead to the shaft.

Sumps were constructed on both sides of the safety bulkhead, that on the heading or working-chamber side to provide a rapid flow of water from the face to the drainage capacity of 90,000 gal. (Fig. 3). A pumping station, comprising four pumps having a combined capacity of 7000 gal. per min., was installed at the safety-bulkhead sump to carry the drainage through the 18-in. discharge to the pumps at the shaft bottom.

Emergency Equipment.—A weir provided with an automatic attachment was placed at the sump to indicate the flow of water as work in the face was advanced; in an emergency, this could control the operation of the bulkhead pump station.

FIG. 7.—SAFETY BULKHEAD LOOKING SOUTH TOWARD HEADING.
Heading bulkhead with blanket removed can be seen; also drift *A*, later abandoned. A 73-ft. space intervenes between bulkheads. Taken from pump-station side of safety bulkhead. Observe grouting pipes, removable track section, main ventilating pipe, and other conduit facilities.

At least one attendant was on duty at all times at the safety-bulkhead pump station. Telephones were provided between bulkhead, shaft bottom, and surface. Instructions as to steps to take in an emergency were painted in large letters on a board placed at the bulkhead.

An incandescent lighting system was used for illumination, and workmen carried flashlights, so that they would have enough light to permit them to escape in case of power failure or other emergencies, such as closing of the bulkhead doors, manipulation of the valves and other safety accessories. A storage-battery unit, which could provide ample lighting, was installed at the bulkhead. This equipment was tested at least once each shift. The emergency lighting system consisted of a 6-volt, 780-amp-hr. storage-battery unit of the type commonly used for a storage-battery locomotive connected to a 6-volt, 15-candlepower, automobile head lamp. A small rectifier was provided near the safety bulkhead to ensure a constant charge to the battery.

A tugger hoist was employed to close the safety door rapidly. In addition, a storage-battery locomotive was kept in constant attendance and readiness to close the safety door by a cable should the compressed-air system fail.

Advancing the Drifts

After safety bulkhead and pumping facilities had been provided, the blanket was removed to the face of the heading bulkhead and an attempt made to advance the face by a side drift,[4] but this had to be abandoned and filled with concrete.

A new drift *B* (Figs. 3, 6 and 9), circular in section, concrete-lined to a finished diameter of 7 ft., with a minimum thickness of 18 in. of concrete, was begun at the bulkhead at the end of the first section of lining, and at the same time through all flowing pipes in the concrete plug of drift *A*, effectively reducing the leakage from this

FIG. 8.—SAFETY BULKHEAD AND SAFETY DOORS, LOOKING NORTH FROM WITHIN WORKING CHAMBER TOWARD SHAFT.

Observe removable track section, removal section of main ventilating pipe, together with flap valve and rope attached; safety door in two sections, each with cable attached; also rubber gasket in doorframe.

tunnel-heading bulkhead on Dec. 17, 1940. It was driven 220 ft. and completed on March 22, 1941. This drift was driven with its axis on the tunnel center line 3 ft. below the spring line; it sloped upward on a 1 per cent grade for drainage. To ensure a safe working place, an efficient seal was provided with the heading bulkhead, and an additional concrete cutoff wall was constructed to tie in to sound rock below.

The procedure followed with drift *B* was to advance the drift heading in 20-ft. stretches, allowing (in each instance) 5 ft. of unlined rock ahead for bulkhead construction against the face if required for grouting in advance of the drift. Thorough grouting was effected through a heading entire area to 15 gal. per min. In advancing drift *B*, a solid concrete barricade was provided at all times against water inflows except in the advance in the heading. Five separate heading bulkheads were used as the small drift advanced and each time grouting was employed. The fifth and final bulkhead was built 220 ft. from the drift portal. Exploratory core boring, as well as the final grouting, proceeded from this point.

A 36-in.-gauge track was laid on the finished invert, and mucking and loading were done by hand into regular muck cars cut to half height for clearance. Two drills mounted on columns were used and the

rounds determined by the nature of the rock.

The concrete was mixed in a 10-cu. ft. Ransome mixer and deposited with a Ransome concrete gun. Invert and arch were poured separately in 20-ft. lengths, with wooden forms; 18 by ⅛-in. steel water stops were incorporated in all construction joints, to prevent water or grout leakage. The standard mix of 2 bbl. of cement per cubic yard was used, adding 2 per cent by weight of calcium chloride to obtain a higher early strength against grouting pressures.

Drainage was provided in the drift by incorporating a continuous 16-in. steel pipe in the invert concrete, which allowed the water encountered in advancing ahead of the lining to drain without running over the completed invert.

Ventilation in the main tunnel was by continuous exhaust, with the end of the vent pipe inbye the safety bulkhead. Ventilation in the drift was provided by a small blower set placed outbye the safety bulkhead and an 8-in. spiral welded pipe carried in the drift.

A series of 3-in. pipes was embedded radially in the lining (Figs. 6 and 9) in rings 4 ft. apart with 14 pipes to the ring and in contact with the rock walls. These pipes were utilized for low-pressure grouting (300 lb. per sq. in.), as the lining was advanced and at the same time provided for radial drilling and consolidation with grout of the entire area that had eventually to be excavated.

Drilling and grouting through the radial holes were undertaken when drift *B* was completed, the roof sufficiently consolidated, and the water shut off so that the full face enlargement could be begun on June 17, 1941.

Pressures in grouting ranged from 300 to 1000 lb. per sq. in., the static water pressure being 290 lb. per sq. in. Approximately 20,000 bbl. (about 80,000 bags) of cement was used for grouting, including 5450 bbl. of high early strength.

Advancing the Main Bore

Tunneling in the main bore consisted of advancing the heading by removing the concrete and consolidated material around

FIG. 9.—LOOKING SOUTH FROM PORTAL OF DRIFT *B*, THE 7-FOOT CONCRETE-LINED DRIFT SUCCESSFULLY DRIVEN FOR 220 FEET THROUGH WATER-BEARING ROCK. DRIFT *A* HAS BEEN FILLED WITH CONCRETE.

Observe particularly grout connections for radially drilled holes in lining; also ventilating pipe. After consolidation by grouting of the mass of rock surrounding this drift, the entire section was removed to the full size of the main bore. The tunnel will be lined with a special steel-shell reinforced-concrete lining through this section and for a short distance on both ends of drift shown.

the drift to full section and then proceeding to remove the rock intervening between shafts 2 and 2*a*. Tunneling was continued

uneventfully, and the south heading of shaft 2 holed through to the north heading of shaft *2a* on Sept. 17, 1941.

After the full-face excavation was completed, the grouting was found to have been so successful that the total water infiltration throughout the full 500 ft. of faulted area was only about 1800 gal. per minute.

This effective grouting was due, probably, not only to the extremely numerous grout connections available but also to the method of washing out the grout holes with water under pressure whenever a hole refused to take further grout.

Another effective agent was the use of high-early-strength cement, which was employed for grouting, under the direction of Fred W. Stiefel, chief engineer for the contractor. The high-early-strength cement, which was much finer (about 400 mesh) than the regular cement, appears to have been more effective in cementing the finer interstices in the rock formation. Furthermore, as shown by actual experiment with various cements, the shrinkage in bulk of the high-early-strength cement was much less than that of the regular cement. Numerous tests with various cements showed conclusively that a given quantity of high-early-strength cement would result in a bulk of hardened grout approximately twice as great as that of an equal quantity of the regular cement.

Conclusions

The success achieved in driving the section of tunnel described in this report indicates that the method used in driving a tunnel safely through fractured hard rock carrying water under high pressures was practical and efficient.

The authors believe that further experiments in the use of high-early-strength cement offer a fruitful field for investigation and that there is every reason to believe that its use in grouting will be more extensive.

The main conclusion that can be drawn from the description of this very commendable engineering feat is that it is possible to prosecute a dangerous and difficult piece of tunnel work and still maintain full control of conditions. Although probably this tunnel face could have been opened without grouting, uncertainty existed as to the quantity of water to be handled, and installation of pumping facilities, the adequacy of which would be problematical, would have entailed prohibitive expense.

Furthermore, an inrush of water could be disastrous. As it was, by using a safety bulkhead, advancing a small controlled and concrete-lined-drift as rapidly as pumping capacity would permit, and installing a pumping plant of reasonable size, it was only a matter of time until the tunnel was completed, and no undue chances had to be taken.

Acknowledgments

For assistance in obtaining material and preparing this report, the authors are indebted to Fred W. Stiefel, Chief Engineer, A. T. Araneo, Shaft Engineer, and C. S. Young, Safety Engineer for Samuel R. Rosoff, Ltd., contractor on Contract No. 313, Delaware aqueduct; to Charles M. Clark, Chief Engineer, N. C. Haldridge, Department Engineer, Max F. Freund, Division Engineer, A. E. Hilliard, Section Engineer and R. St. Leger, Shaft Engineer of the New York Board of Water Supply on the Delaware aqueduct tunnels; and to L. L. Naus, Engineer with the Bureau of Mines.

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Trucking Operations at New Cornelia Mine

By Harry H. Angst,* Member A.I.M.E.

(New York Meeting, February 1942)

The history and efficiency of 40-ton capacity dump trucks for surface waste removal at the New Cornelia opencut copper mine, at Ajo, Ariz., are summed up in this paper. Tabulations of truck performance, together with detailed cost figures, prove the value of truck transportation for certain special operations at a large mine. Emphasis is given to the fact that truck transportation not only reduces the haulage costs but also reduces the breaking ground and shovel-loading costs per ton.

The New Cornelia opencut copper mine is in the desert. The climate is ideal for seven months of the year, but the five summer months are uncomfortably hot. The elevation is 1820 ft. above sea level and the annual rainfall is approximately 6 inches.

The ore body is a monzonite-porphyry deposit. The waste-rock capping consists of hard rhyolite, conglomerate and porphyry. All the material is hard and cannot be excavated without blasting. The rhyolite and conglomerate waste is very blocky. The weight of the rock in place is 2.16 tons per cubic yard.

The average mine production per day is 22,000 tons of ore and 18,000 tons of waste rock, the latter consisting of surface rock and material within the ore body too low in grade to send to the concentrator. The rock is loaded by electric shovels with 4½-cu. yd. dippers. The primary transportation of ore to the concentrator and waste to the dumps is by 30-yd. dump cars and oil-burning steam locomotives; therefore truck haulage of waste is a minor factor of the New Cornelia operation. However, as New Cornelia was one of the first mines to use large-capacity dump trucks for haulage, the results obtained are of interest.

Equipment

On the south side of the New Cornelia is a hill rising about 500 ft. above the normal surface elevation, roughly 1000 ft. in diameter at its base, called locally Arkansas Mountain peak. Exploratory drilling in this area developed additional ore that could be made accessible for open pit operation only by moving the mountain face a maximum distance of 400 ft. to the south. The slope of the mountain adjacent to the pit averaged 35°, and the small area involved precluded the use of train haulage for the removal of the 4,000,000 tons of waste estimated above the normal pit elevation. A study of dump-truck haulage at Boulder Dam and San Gabriel Dam led to the decision to use large-capacity end-dump trucks rather than smaller trucks, for the following reasons:

1. The truck would have to withstand the impact of hard, heavy chunks, some of them weighing as much as 10 tons, which would be dumped from the 4½-cu. yd. dipper.

2. The coarse, rocky, abrasive material would soon damage a light truck body.

3. The body of a small truck could not be efficiently loaded with coarse rock.

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4. Road conditions were favorable for heavy loads.

5. The average ½-mile haul to the dump would be short, so the speed factor of the small truck was not necessary.

6. The anticipated level haul did not require excessive motive power in a large truck.

Four trucks of the following specifications were purchased:

End-dump tub body of 22½ cu. yd. capacity.
Body length, 16 ft.; body width, 8 ft. 6 in. front, 10 ft. rear; body height, 62½ in.
Wheel base, 152 in. plus 56 in. to rear tire.
Turning radius, 40 ft.
Rear tires, 13.50 × 24 (eight).
Front tires, 12.75 × 24 (two).
Tare weight, 40,000 lb. (20 tons).
Load capacity, 80,000 lb. (40 tons).
Clearance of rear dual tires, 112 in.
Gasoline motor, 130 hp.
Five-speed transmission.
Speed, miles per hour: 20, high speed, empty; 15, maximum load on level; 3, up 6 per cent grade, loaded (low gear).
Gar Wood hoists.

Spare parts purchased were: one truck body, one transmission, one differential, one front axle.

The equipment on the truck job is as follows:

1 electric shovel: full revolving; weight 200 tons; caterpillar traction; 50-ton bail pull. Digging radius, 46 ft.; digging height, 32 ft. Dipper, 4½ cu. yd.; 1½-in. hoist cable. (The same shovel has loaded continuously on the truck job with no spare shovel.) Four dump trucks, one sprinkler truck, one service truck, one R.D.8 bulldozer, one repair garage.

The haulage operation crew per shift consists of:

Four truck drivers.
One sprinkler-truck driver.
One service-truck driver.
Two truck mechanics at repair garage.
One bulldozer operator.
Two roadmen.
One dumpman.

Truck crews are allowed 15 min. at beginning of shift for oiling and 15 min. for lunch out of the 8 hr. operating shift. The shovel is idle during these periods.

Truck Performance

The trucks were put into operation April 1, 1937, and are still in use. The haulage of material on Arkansas Mountain with trucks was so successful that it was found advisable to use them for waste haulage from other portions of the mine where the haulage distance was not excessive and where the grade adverse to the loads would not slow up the speed enough to necessitate purchase of an additional truck unit. Table 1 shows truck performance.

Tires

Tire wear is a big problem because of the hard, rocky character of the material handled. The main haulage road to the waste dump is surfaced with leach tailings, but the shovel area and dump area are covered with sharp rocks, which damage the tires. The sharp turns at shovel and dump cause rear tires to slide over these rocks, and there is frequent cutting of tires. Tire records are as follows:

Tons hauled per set of tires, 338,375 average.
Miles per set of tires, 8,300 average.

Operating Cycles

Cycle when hauling ½ mile to waste dump on level:

Loading time, 2 min., 30 sec.
Trip to dump, 1 min., 57 sec.
Dumping, 18 sec.
Return to shovel and spot, 1 min., 46 sec.
Total: 6 min., 31 sec. (requires 3 trucks).

Cycle when hauling 2200 ft.—1200 ft. level, 400 ft. on 6 per cent grade, and 600 ft. on 3½ per cent grade:

Loading time, 2 min., 30 sec.
Trip to dump (2200 ft.), 3 min., 36 sec.
Dumping, 20 sec.
Return to shovel and spot, 1 min., 46 sec.
Total: 8 min., 12 sec. (requires 4 trucks).

Cycle when hauling 4000 ft. to waste dump—1000 ft. level, 1200 ft. on 3 per cent and 1800 ft. on 2 per cent grade:

Loading time, 2 min., 30 sec.
Trip to dump, 8 min., 40 sec
Dumping time, 20 sec.
Return to shovel and spot, 4 min., 30 sec.
Total: 16 min.

Four trucks cannot keep the shovel busy. The average number of trips per truck per shift dropped to 24.

Cycle when hauling 3600 ft. to waste dump on 2 per cent grade:

Loading time, 2 min., 30 sec.
Trip to dump, 5 min., 10 sec.
Dumping time, 30 sec.
Return to shovel and spot, 3 min., 50 sec.
Total: 10 min.

Four trucks can keep the shovel busy loading 4500 tons per shovel-shift.

TABLE 1.—*Truck Performance to October 1, 1940*

Item	1937, 9 Months	1938, 11 Months	1939, 11 Months	1940, 8½ Months	Total
Number days shovel operated	259	300	309	202	1,070
Shovel-shifts operated	610.00	701.87	921.52	497.28	2,730.67
Tons per shovel-shift	3,091	3,448	3,785	3,711	3,530
Tons of waste hauled	1,885,372	2,420,073	3,488,400	1,845,200	9,639,045
Average number trucks in use	3.08	3.00	3.38	3.34	3.21
Truck-shifts operated	1,878.63	2,106.46	3,116.81	1,663.08	8,764.98
Truck-hours operated	15,029	16,852	24,934	13,305	70,120
Average hours per truck					17,530
Tons hauled per truck-shift	1,003.5	1,148.9	1,172.1	1,109.5	1,099.7
Total number truck trips	56,368	62,850	87,210	46,130	252,558
Tons per trip	33.45	38.51	40.00	40.00	38.17
Average number trips per truck-shift	30	29	29	28	29
Miles per loaded trip	0.183	0.284	0.645	0.438	0.424
Miles per empty trip	0.183	0.284	0.645	0.438	0.424
Miles per total trip	0.366	0.568	1.290	0.876	0.848
Truck-miles loaded	10,303	21,095.5	57,661	20,205	109,264.5
Truck-miles empty	10,303	21,095.5	57,661	20,205	109,264.5
Truck-miles total	20,606	42,191	115,322	40,410	218,529
Net ton-miles of material	344,149	687,097	2,250,225	808,374	4,089,845
Average lift, ft	0	25	78	40	
Truck availability, per cent	73.83	81.23	81.04	80.91	79.52

Individual Truck Record

Truck No.	Tons Hauled	Hours Operated
1	2,396,274	17,502
2	2,504,212	18,117
3	2,443,307	17,660
4	2,295,252	16,841
Average	2,409,761	17,530

Operating Data of Truck Job

The shovel operates on a 50-ft. bank, with wide berm to permit turning of trucks at the shovel. The shovel does not move parallel to the bank face, but heads in at about a 45° angle, which permits the spotting of a truck on both sides of the shovel and requires a maximum shovel swing of 90°.

The blasting of a 50-ft. bank throws the broken material approximately 100 ft. A bulldozer pushes the splattered rock against the broken mass. The shovel thus starts loading in a bank several feet high. As the digging radius of these shovels is 45 ft., there is room to spot the truck. All digging is on a level bench. The shovel continues heading into the cut until the bank face is cleaned of all broken material. No broken ground is left as a cushion when shooting a solid bank at New Cornelia, as better breaking efficiency is obtained by shooting a free and straightened bank. The slope of the bank when cleaned is usually ½ to 1 in this rock.

Overhead power-cable supports are used to permit trucks and bulldozer to move freely without interfering with the trailer cable to the shovel. These supports are used throughout the mine where shovel and churn-drill cables cross a roadway.

The shovel loads into the truck any material that will go through the 4½-yd. dipper. Fine material is first dumped into the bottom of the truck as a bed for extremely large chunks. Some of the latter weigh as much as 10 tons, but the average weight of the large boulders is 5 tons.

The trucks back into the spotting position. Drivers leave the truck while it is being loaded because of the danger of flying rocks. No loading is done over the front end of the truck. The first dipperful of material is dumped at the rear end and dumping is continued until the last dipperful is placed at the front end. The average number of dippers per truck is 5½. As the trucks are loaded to body capacity, there is some spillage from the open rear end, especially when hauling up grades adverse to the loads.

The driver does not climb on to his loaded truck until a signal is given by the shovel operator. Also, he does not move the loaded truck until the pitman has removed all rocks from the ground in front of the tires and has given him an O.K. signal. The truckman uses an air whistle on the truck and the pitman and dumpman use police whistles for signalling.

At the dump, the truck turns around and backs to the dump position. One man (a Papago Indian) handles the dump. He signals the driver where to dump the load and stops the truck at the proper spot for dumping. Although 250,000 truck loads of rock have been dumped, no truck has ever gone over the dump. Dumps have a maximum height of 275 ft., but no trouble has been caused by settling, which indicates the physical character of the material handled.

One bulldozer is used continuously for leveling around the shovel for leveling off spillage on the main roadway and for leveling off the dumps. The dump area is so large that dumping can be done for half a shift before the bulldozer is required.

The roadways are about 30 ft. wide, to permit trucks to pass at speed. Roadmen keep the surface clear of spillage. The roads are lighted at night by electric lights spaced at 100-ft. intervals along one side. Two floodlights are used at the dump site. A sprinkler truck prevents excessive dust from these large trucks. As the driver is on the left side of the truck, the left-hand side of the road is used. This allows the driver to watch the outside of the road on embankments and thus avoid running off.

The service truck supplies fuel and lubricants to the trucks at the place of operation, and is used to keep the proper air pressure in the tires. Tires carry 95 lb. of air. Trucks are completely serviced each day, the average time consumed in servicing a truck is 30 min. On a two-shift operation, the servicing is done on the off shift; on a three-shift operation, the trucks are serviced on day shift.

The usual strict safety program of the Phelps Dodge Corporation has been followed for trucking operations. A Code of Safe Practice for Truck Drivers was adopted, and no person has suffered a lost-time accident from truck haulage in 3½ years.

Truck Changes Made

Truck delays due to improper design, poor material and poor workmanship necessitated some major changes in the trucks as follows:

1. The flat wearing strips on the floor of the body were changed to ½ × ½ × 3-in. angles. These angles made a much stronger floor and offered no impediment to the flow of the rock when dumping.

2. The flat canopy over the driver was changed to a gable type and the front body end was raised 16 in. This canopy is at an

TABLE 2.—*Normal Operating Costs*

	Per Shift				Per Ton			
	1937	1938	1939	Diesel 1940	1937	1938	1939	Diesel 1940
Operating:								
Labor	$ 5.38	$ 5.64	$ 5.98	$ 5.95	$0.0054	$0.0045	$0.0053	$0.0053
Fuel	2.51	3.65	4.35	1.25	0.0025	0.0029	0.0039	0.0011
Tires	4.52	8.31	6.66	6.71	0.0045	0.0066	0.0060	0.0059
Supplies	3.66	2.79	3.35	1.90	0.0036	0.0022	0.0030	0.0017
Total	$16.07	$20.39	$20.34	$15.81	$0.0160	$0.0162	$0.0182	$0.0140
Repair and maintenance:								
Labor	$ 2.93	$ 7.90	$ 6.18	$ 9.67	$0.0029	$0.0063	$0.0055	$0.0086
Supplies	3.66	11.40	10.25	7.10	0.0036	0.0091	0.0092	0.0063
Total	$ 6.59	$19.30	$16.43	$16.77	$0.0065	$0.0154	$0.0147	$0.0149
Grand total	$22.66	$39.69	$36.77	$32.58	$0.0225	$0.0316	$0.0329	$0.0289

angle that sheds rock spilled by the dipper. The raising of the front end permitted faster loading and increased the body capacity by several tons.

3. The front of the body was reinforced by horizontal bands of 40-lb. rail.

4. The front axles were made heavier and of improved material.

5. The truck frames were too weak, so much heavier frames were installed.

6. Splined jackshafts replaced the original round-keyed shafts.

7. The ½-in. leaves of the rear springs were changed to ⅝ inch.

8. The original 5-gal. radiators were too small for the extreme heat at Ajo, so 7-gal. radiators were installed.

9. The 130-hp. gasoline motors were removed after 15,616-hr. average service, hauling 2,139,971 tons of material each. Cummins Diesel motors of 150 hp. were installed.

The trucks are still in good condition and in continuous use. The original bodies are still in fair shape after hauling 2,500,000 tons each. As shown in Table 1, the truck delays were excessive and amounted to 20 per cent of the possible operating time.

The truck speed, particularly on grades, has been increased slightly with Diesels. The tons hauled per gallon of Diesel fuel is double that of gasoline. Motor delays have been reduced to a minimum with Diesels.

COSTS

Normal operating costs per shift and per ton for the New Cornelia truck operations are shown for gasoline and Diesel-powered trucks in Table 2.

CONCLUSIONS

Truck haulage at New Cornelia has been a success for special operations.

The Diesel motors are far superior to gasoline motors in haulage operations of this type, both as to efficiency and costs for operating.

Table 3 shows a comparison of gasoline and Diesel motors on the same haulage from September 1939 to April 1940.

TABLE 3.—*Comparison of Fuels*

Haulage Data	Gasoline	Diesel
Gallons fuel per truck-shift	42.9	19.0
Tons hauled per gallon fuel	27.3	55.0
Ton-miles per gallon fuel	13.6	27.5
Gallons fuel per ton hauled	0.039	0.018
Gallons motor oil per truck-shift	0.0092	0.0092
Gallons hoist oil per truck-shift	0.0035	0.0035
Pound grease per truck-shift	0.0185	0.0185
Motor availability—per cent	92.60	96.13

Tire costs are excessive on the large trucks. Suitable tires of this size have not yet been developed for carrying such heavy loads (60 tons).

Good road maintenance is essential.

Truck-maintenance costs are excessive as compared to operating costs on this type of truck

The life of these trucks will average at least 25,000 hr. of running time.

Haulage by truck not only reduces the haulage costs but also reduces the costs for breaking ground and loading.

When loading into trucks with a short swing, the shovel-loading speed is greatly increased over that of a swing 90° to 180° when loading a train of cars. At New Cornelia this increase in tons per shovel-shift is more than 500 tons. Also, the shovel is not tied up when it encounters material it cannot dig. With truck haulage, the shovel can quickly move to broken ground and the drilling and blasting will not interfere. With train haulage, the shovel must stop until the obstructing rock is drilled and blasted.

In digging a rock bank, loading trucks, the shovel digs all the blasted material and "faces up" the bank perfectly. But when loading into dump cars, the shovel-digging distance is limited by the distance from the loading track. As it does not pay to line the track for a cut of 5 or 10 ft. of broken rock, this already fragmented material is again shot by the succeeding blast. Costs for breaking ground in the same character of material have been reduced by 20 per cent where truck haulage took the place of train haulage.

Since the introduction of dump trucks at New Cornelia, the Morenci operations have moved 30,000,000 tons of waste rock with the same type of dump truck. The trucking there, however, was a special job of developing the mine for later train operation.

The adaptability of truck haulage for many rock-moving operations has made these trucks very valuable at New Cornelia mine. Many uses have been found which were not contemplated when the trucks were purchased.

The trucks have been used in building new waste dumps for later train haulage instead of building dump trestles or raising tracks. By using the trucks, a dump can be built economically in a very short time.

The construction of a storage yard for ore trains proved to be a simple job by the use of truck haulage.

It is planned to use truck haulage for the next "drop cut" in lowering the pit bottom another 40 ft. to establish a track-operating bench. When using trains, the 4½-yd. shovel must make four separate cuts, each about 10 ft. deep, to arrive at the 40-ft. depth. This necessitates "block and block" loading with shovels as well as excessive track laying and moving. With trucks, hauling to a ramp, the ore can be dumped directly into cars and the shovel can dig on the final 2 per cent grade to the ultimate 40-ft. depth. A saving is expected because of faster shovel loading and more efficient blasting, with no track work until the drop cut is completed. It is estimated that with truck haulage a drop cut that requires the movement of 500,000 tons of ore can be made in less than one-half the time required with train haulage.

These uses illustrate a few of the numerous operations around a large mine to which truck haulage is adaptable.

The New Cornelia mine will continue to use these trucks until they are finally worn out. Present plans are to haul another 7,000,000 tons from certain pit-waste areas, where train haulage had been used. The maximum lift will not be more than 100 ft., and maximum haul will be one mile. Under these conditions the truck haulage will be more economical than train haulage.

Development of Tractor and Airplane Transportation in Manitoba

By George E. Cole*

(New York Meeting, February 1940)

While many parts of Canada's pre-Cambrian shield are well served by railway, it is frequently necessary for prospecting purposes to proceed farther into areas inaccessible by rail. To such areas in summer the canoe was at one time the accepted means of travel for the prospector while the dog team was much used in winter. Although summer travel was much improved by the development of the outboard motor, the lack of facilities for cheap transportation has been a serious hindrance to the employment of capital in opening up the distant fields.

The pre-Cambrian surface of Manitoba, like that of Ontario, is hummocky, with rounded hills and ridges of rock alternating with basinlike depressions and valleys occupied by lakes and rivers or by many low-lying portions covered with muskeg. Routes to the mines have not usually been direct or easy and seasonal vagaries have presented problems in themselves.

In summer, water transportation may be the cheapest method for the longer distances but it is not every water route that will satisfy the conditions of loading freight on to a boat or scow and of conveying it without interruption to its destination. By its location, a mine may require a large program of boat transportation in summer over part of its route, to be followed by winter freighting for the remainder. In order to carry on development and production over a long period, it may be necessary to haul large quantities of supplies during the winter. This seasonal problem, particularly for the out-of-the-way properties, is so important and demanding as to make it a major operation in the life of the mine. The use of the tractor during winter months for heavy machinery and bulk supplies, and the use of the airplane throughout the year for the transportation of personnel and for emergency requirements, have greatly improved transportation to places that are inaccessible by rail. Canada has pioneered in both these methods.

Development of Transportation

Twenty-two years have seen a marked change in transportation methods in Canada's mining country. Horse-drawn sleighs have given way to the tractor and that to some extent has been supplanted by the airplane.

One of the earliest ventures involving transportation of considerable magnitude was that of the Mandy mine, on Schist Lake in northern Manitoba. According to accounts of early mining activities in Manitoba, the Mandy mine, beginning December 1916, had 3300 tons of ore hauled 40 miles by sleigh to Saskatchewan River and in summer this was conveyed by barge to The Pas, 130 miles, and then by railroad to Trail, British Columbia, 1200 miles.

The mine shipped 6000 tons of ore in 1918 and 15,000 tons in 1919. In winter of the latter year, as many as 300 teams of horses were used on the 40-mile stretch to

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navigation. The average load of a single team was 6½ tons and the cost was 37½¢ per ton-mile.

The records show that: "In 1917, a gas tractor of the caterpillar type, capable of hauling 20 tons, was being tried out over the portages." This marks the first use of a tractor in the mining fields of Canada's northern areas.

In 1925, following the discovery of the Howey mine at Red Lake, northwestern Ontario, tractors were tried out between Hudson, on the Canadian National Railway, and the property but owing to slush on the lake the attempt at hauling was a failure. Even a year later, at Flin Flon, there was still uncertainty as to the economy of using the tractor for winter haulage and horses were used for freighting between The Pas and the mine. In the same winter, freighting was done with horses from The Pas to Cold Lake, 140 miles. For further development, 2600 tons of machinery and supplies were hauled to the property in 1928 from mile 55 east of The Pas on the Hudson Bay Railway. Hauling commenced Jan. 10 with 150 teams of horses and sleighs. Later it was feared that the winter "break-up" would find some freight still en route. Three Holts and one Linn tractor were pressed into service. All freighting was done in 90 days. The tractors, however, did not reach their maximum efficiency because the roads had not been well prepared for them. Cost per ton-mile for the winter's work was 56 cents.

Flin Flon and Island Falls

Early in 1928, the option on the Flin Flon mine was exercised. In a direct line 56 miles from the mine a power site was available at Island Falls, on Churchill River. This presented a particular freighting problem where success depended on efficiency. The road would be used during two winters at least, and later if the power-plant capacity had to be increased further transportation of heavy material would be necessary. To meet conditions for heavy loads, speed of travel, and low cost of equipment maintenance, the road had to be provided with a good bottom and then iced to permit of the hauling of double the load that could be taken over a snow road.

In summer the land-road sections (portages) were surveyed, the right-of-way cleared of stumps and stones and leveled to a uniform surface. Sharp grades were avoided by cuts and fills and the few curves necessary were set out with as long a radius as possible. The land road in general conformed to the requirements of a good summer road. With transshipping yards built near Flin Flon plans were made for winter haulage on a scale hitherto not attempted in northern areas. This involved the movement of a large tonnage of freight, 25,000 tons, over a fairly long road, 69 miles, in a comparatively short time, 90 days.

What is known as the "freeze-up" in northern areas usually comes early in November, while the "break-up" of the winter season comes about the first of May following. It is a decided advantage to have ice well formed before the first snow fall but this does not often happen. The formation of ice can be greatly accelerated by keeping the snow plowed off as soon as ice is thick enough to carry a team of horses. Usually this thickness, 6 in., is attained about the middle of December and by the end of the month heavy loads may be hauled. Ice 18 in. thick is sufficient for medium-loaded tractor trains while 24 in. is desirable for heavy loads. For good ice making a temperature of at least 20°F. below zero is desirable. As temperatures of 40° to 50°F. below often prevail in the Flin Flon area, ice reaches a thickness of 36 inches.

Preparations for the winter road were commenced as soon as ice was formed on the lakes. Snow falling to a depth of several inches on the land sections was rolled and iced while the lake sections were kept clear

of snow for a time. (The snow fall in the area averages about 35 in.) Great care was needed where the land road entered or left a lake, for at those points the ice is usually weak. In general the lake road was some distance away from the shores of the lake. After every snowstorm it was necessary to plow the road well beyond the width of the actual roadway. The accumulation of snow at any time on lakes tends to form slush, which, as it does not freeze if there is snow on top of the water, makes the road dangerous. Plows were attached to the tractors and were operated when necessary.

A base camp maintained at the transshipping yards near Flin Flon included office and staff house, bunkhouse and cookhouse, repair shop and blacksmith shop, warehouse and platform, garage and barn, powerhouse and oilhouse.

Small camps, each including bunkhouse, cookery and stable, were maintained at several places on the portages for crews working on the road, and through these camps telephone communication was provided.

At the Island Falls terminal, a third and smaller camp included bunkhouse, cookery and garage for the switching and unloading crews.

In the work of transportation, switching and maintenance of the road the following equipment was used:

Twelve 100-hp. Linn tractors with snow plows attached,
One hundred and fifty special 60-in. gauge sleighs with racks,
Ten large and four small cabooses,
One 60-hp. tractor for making up loaded trains at transshipping point,
Two 30-hp. tractors at the terminal for switching,
Fifteen teams,
Four horse-drawn snow plows,
Seven tank sleighs and water tanks.

Together with these, there were two small service trucks, a snowmobile and an automobile.

With the road camps and equipment ready, the program of unloading and transshipping at the base, movement to and unloading of freight at the terminal was carried out on schedule. Tractors and loads were dispatched and kept track of en route. A train consisting of four to six sleighs, including a caboose, was kept in continuous movement on the road, stopping only for inspection, oiling or mishaps. The crew consisted of two drivers and two helpers operating in pairs on either 6-hr. or 8-hr. shifts. The crew slept in the caboose and meals were prepared and eaten while the train was in motion.

The average time taken for the 69-mile trip with loads was 23 hr. and 2¼ hr. was spent at the terminals. The load of a sleigh varied from 15 to 38 tons and while the average load of a train was 75 tons the maximum reached 120 tons. The return trip to the base averaged 14¼ hr. with speeds between 4 and 7 miles per hour. At the base 6 hr. was allowed for reconditioning.

Between Dec. 15 and March 15, the amount of freight moved was 22,500 tons, of which 20,000 tons was hauled in 60 days. In a second winter's freighting (1929–1930) another 12,650 tons of freight was hauled, so that with all equipment on the ground at Island Falls, the hydroelectric plant was ready to supply power at Flin Flon by July 1930.

The cost of building the road was about $7000 a mile and maintenance $300 a mile. With gasoline at 22½¢ a gallon (Canadian) the cost of freighting, exclusive of depreciation on equipment, was about 22¢ per ton-mile. The cost of the equipment for hauling, and maintenance of roads was $202,000.

Sherritt Gordon Mine

The railway built to Flin Flon made possible a shorter haul to the Sherritt Gordon mine than that of the previous winter. Although a branch railway line from Cranberry portage to Sherridon was in course of construction, a second winter's

freighting was necessary in order to get a new shaft and a pilot mill under way. A road 54 miles long was cut from Cranberry portage to the mine, 35 miles being over frozen lakes and the remaining 19 miles over 27 portages. The road over each portage was kept well iced, and the lakes were plowed after each snowfall.

FIG. 1.—SOME DIFFICULTIES IN TRACTOR HAULAGE OVER FROZEN SURFACES OF NORTHERN LAKES.

Hauling was done by Linn tractors and trains of sleighs, including a caboose for each train. At the railway and the mine trains were delayed only long enough for loading and unloading. The routine kept one train loading, a second on the road and a third unloading.

A total of 3600 tons of freight was hauled in a period of 90 days at a cost of 27¢ per ton-mile and it was estimated that this could have been reduced if tonnage of freight had been sufficient to justify an extra expenditure in grading the portages to make possible the hauling of heavier loads.

Island Lake Mine

The tractor as a means of winter transport over comparatively long distances was definitely settled in the winters of 1929 and 1930. In the winter of 1933 the mine and mill equipment for the Island Lake mine, shipped the previous summer by boat to Norway House, at the north end of Lake Winnipeg, were transported 175 miles over winter road to the mine. The hauling was done by a contractor who was paid 25¢ a ton-mile. Much difficulty was caused by heavy snowfalls before the lakes and muskegs were sufficiently frozen. Weather that followed was exceptionally severe, with strong winds blowing constantly from the northwest and temperatures during the freighting period averaging 40°F. below zero. In spite of the intense cold, there was not a single tractor employed in the work that had not, at one time or another, been immersed in either the water or the muskeg. (Fig. 1.)

During the hauling, two jaw crushers were lost through the ice. The first of these was recovered when a diver, flown to the scene, finding that the crusher had disappeared entirely at a depth of 22 ft. in the silt and soft mud of the stream bed, fastened grappling irons to the crusher, which was raised to the surface with blocks and eventually transported to the mine.

God's Lake and Sachigo River Mines

The next property to benefit by the use of a tractor for winter haulage was that of God's Lake Gold Mines Limited. The nearest railway facilities were at Ilford, on the Hudson Bay Railway, 132 miles away. In the winter of 1933–1934 a road was cut from the railway to the mine and 1260 tons of freight, including a complete mining plant and supplies for one year's operations, were hauled to the mine.

During the first three months of 1935, tractors hauled 4713 tons of machinery, building material and supplies over this

winter road, amounting in all to 668,600 ton-miles, for which the over-all cost, exclusive of depreciation, was 22.85¢ per ton-mile.

In 1936–1937, besides the 2803 tons required for the God's Lake mine and its power plant, a considerable tonnage was hauled another 120 miles to the east for the development of a mine at Sachigo River, in Ontario. With the experience gained on the road from Ilford to God's Lake, the addition of another 120 miles freighting from God's Lake to Sachigo River was successfully accomplished in the winter of 1937–1938. Even with adverse conditions, 1350 tons of freight was easily hauled to God's Lake and 1150 tons to Sachigo River. The God's Lake Gold Mines Limited contracts the hauling of freight to Sachigo River at $60 a ton.

The route from Ilford to God's Lake passes over 48 lakes in a distance of 54 miles and over 50 portages in 78 miles, while the route from God's Lake to Sachigo River is over 31 lakes in 78 miles and 34 portages in 42 miles. As this winter freighting by tractor from Ilford to Sachigo River via God's Lake is one of the longest into a mine area, a digression is made to discuss some of the organization, the practical technique in the operation and experience developed in the work.

The tractor is now affectionately referred to in the mining areas as a "cat," obviously the diminutive of caterpillar. Two tractors, eight heavy-duty sleighs and a caboose are referred to as a "swing." The trains travel in pairs, so that if the need arises the crews can help one another. If there is a portage with a hump, both tractors can be hitched to haul a train over the rise. The first tractor pushes a snow plow and the second tows the caboose at the end of its train. The crew numbers seven—swing boss, three drivers, two brakemen and a cook. They work in shifts as the "swing" travels day and night until its destination is reached. The work is carried out in 8-hr. or 6-hr. shifts or less, depending on the severity of the weather. The swing load averages 65 tons over the season. The round trip from Ilford to God's Lake occupies four days and to Sachigo River seven and one-half days.

The roads used for winter traffic into these mines may be spoken of as snow roads, as compared with the iced roads used in hauling from Flin Flon to Island Falls. The tonnage of freight being much smaller, the time and expenditure on the roads has of necessity to be cut down.

Usually the trail-breaking loads consist of oil, lumber or other nonperishable freight. If a tractor and load do go through the ice they can usually be recovered. Salvage operations may cost up to $1000.

Berens River Mine

A further development, that of Berens River Mines Limited (Fig. 2) has made use of water and air transportation, and water and tractor transportation. The former will be discussed under the "Airplane." In the latter was involved the handling of 2500 tons of material and equipment for a 225-ton mill and a 2000-hp. power plant for the property at South Trout Lake, in the Favourable Lake area of northwestern Ontario. The freight was moved from Winnipeg within a month after Sept. 19, 1938, via Lake Winnipeg to Berens River Landing, where it awaited a further haul over a winter road that had been cruised and cut during the summer. The route carried over 115 miles on land and the remainder, 65 miles, over lakes, the greatest stretch over water being 40 miles. Five camps were established along the route.

In 1939 for the winter hauling 10 Diesel tractors and 60 heavy-duty sleighs were used. From 40 to 50 men were employed, of whom six or eight were kept as a maintenance crew to look after road upkeep and the testing of ice on the lakes. All freight was delivered to the mine and power site on time. With the mill in opera-

tion, precipitates will be stacked and hauled in winter by tractors on their return to Berens River Landing, or some may be carried at times by airplane at a comparatively low charge.

FIG. 2.—BERENS RIVER MINE, WINTER OF 1938–1939.

The cost of boat freighting from Winnipeg to Berens River Landing was $4 a ton, for the 190 miles.

Rice Lake

Manitoba has but one area—Rice Lake and the San Antonio mine, which is served equally well summer and winter. Tractor trains operate in winter from Pine Falls over a road 42 miles long, while in summer the area is reached by boat to English Brook, thence by smaller craft on the Wanipigow (Hole) River for the remainder of the route.

TRACTOR'S MERITS

The tractor has shown that transportation to isolated mine locations is no longer a serious problem, and that the long severe winters of the northland help rather than delay freighting. The tractor defies extreme temperatures, and while more effective in the northern winters than at other seasons it has proved to be versatile, as it can be turned from one type of work to another.

In operation it gives dependable service at a comparatively low cost, and its maintenance is not excessive. At fair speed its ability to haul heavy loads on sleighs over poor roads is, of course, a feature of its performance. Great care cannot be given to the maintenance of long winter roads, particularly when the tonnage to be hauled is not great. That the tractor provides an excellent power for plowing makes it ideal in clearing snow roads.

It does its best and cheapest work when loaded as nearly as possible to its rated capacity. In winter hauling, then, it can be well loaded and travel easily where roads are good. In hilly stretches the trains can be split into two or more sections, or two tractors can for a time be used on one train.

The cost of freighting depends greatly on the condition of the road, and for that reason the road itself should be located and constructed within economic limits on good engineering principles, yet for the winter transportation the type of road must also depend on the character of the country it traverses.

While today there appears to be a trend toward the use of trucks, particularly where there is a heavy volume of freight, the fact that the truck requires very good

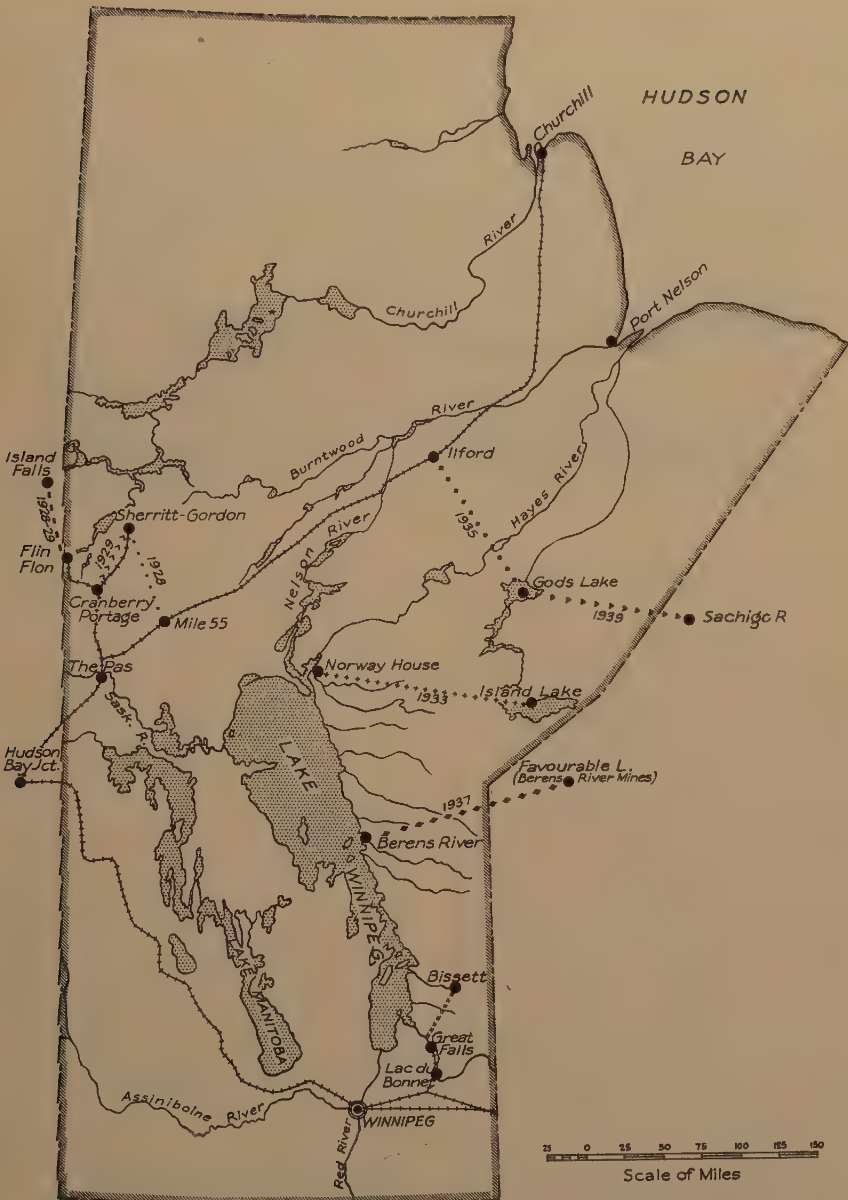

FIG. 3.—ROUTES OVER WHICH FREIGHT HAS BEEN HAULED BY TRACTOR IN THE PROVINCE OF MANITOBA.

roads for its economic operation will militate against its use in the cold, snow-covered, wind-swept northern areas. In the final analysis, the use of the tractor depends on the tonnage to be hauled, the nature of the road to be traveled and the length of the haul, together with winter conditions where great attention cannot be given to maintenance of the road.

Routes over which freight has been hauled by tractor in Manitoba are shown in Fig. 3.

Airplane Freighting

The first airplane to invade the northland of Manitoba left Winnipeg on Oct. 15, 1921, and after making several landings, reached The Pas on Oct. 17, a distance of 325 miles. This machine carried a pilot, an air mechanic and one passenger.

The airplane was first used for freighting in 1925 in the northland, when the Ontario Forestry Service transported 15 tons of freight from Kenora, Ont., to the new gold field at Red Lake just after "freeze-up," as winter setting in early had upset the plans to have the freighting done over a water route. Private air lines followed this venture in 1926 by transporting freight and passengers to Red Lake from both Hudson and Sioux Lookout on Canadian National Railway at a charge of $1 per pound. With experience and competition, this charge was soon reduced. The first aerial transportatiôn in Manitoba was done in 1927 from Cormorant Lake, 42 miles northeast of The Pas on Hudson Bay Railway. Sherritt Gordon Mines Limited had a gasoline-driven diamond drill carried 75 miles from Cormorant Lake to the property at Cold Lake. (This marked the first transportation of a diamond drill in an airplane.) This was followed by the transportation of 40 tons of freight and 42 men, the whole taking 28 days.

Fig. 4.—Airplane used for heavier freight to a field served by railway or water transport.

In writing of this work, E. L. Brown, now general manager, Sherritt Gordon Mines, Limited, says:

> The expense incurred in freighting the supplies and equipment by plane was amply justified by the results. The data obtained enabled us to lay out an extensive exploration campaign for 1928, and to get supplies and equipment for this campaign in time to be freighted-in over a winter road.

By 1929, aerial transportation had emerged from its pioneering stage in northern Canada but the rates were still too high to make it an acceptable method of freighting for heavy machinery and supplies. Even a drastic cut in rates in 1932 did not bring the cost of the plane as a freight carrier within reasonable limits except when the time factor overruled immediate

expense. (Fig. 4.) An instance of the airplane used in this way was that of freighting done by Berens River Mines to its property in the fall of 1936. The fast approaching "break-up" compelled immediate attention because failure of delivery would have cut the mill tonnage in half until such time as the units could be delivered by water.

This contract provided for the transportation of a complete mining plant by airplane. No piece was to be over 2000 lb. in weight or over 10 ft. in length. Most of the equipment had to be knocked down. All freight was shipped before freeze-up via Lake Winnipeg to Berens River Landing. After "freeze-up" some material was shipped by rail to Riverton on the west side of the lake and then hauled by tractor over the ice to Berens River Landing. The work of transportation began late in September and steam plant, sawmill, two tractors, 30 tons of explosives, 40 tons of camp supplies and 50 tons of gasoline and oil, making 350 tons in all, were placed at Berens River Landing ready for the flight to the mine. In addition to this another 40 tons of freight were flown from Lac du Bonnet to the mine, a distance of 225 miles. Included in the transportation were 70 men to work at the mine.

Four aircraft were used in the freighting, flying a distance of 165 miles. To assist in the work radio stations were established at Berens River Landing. South Trout Lake and the mine camp. The speedy operation of freighting permitted shaft sinking to be commenced in January 1937 and by the spring of 1938 sufficient ore was proved to warrant the building of a mill, the supplies and equipment for which were hauled by tractor to the mine in 1938–1939.

To hasten the development plans of the company several months, a further freighting by air was done in July 1938. An 18-ton Diesel tractor with shovel attachment was broken down in Winnipeg and shipped by water to Berens River Landing, whence it was flown 165 miles to the power site at Northwind Lake. The heaviest pieces were the engine, 2400 lb., and the transmission case, with part of the gearing taken out, 2200 lb. The longest piece was the shovel boom, 14 ft. by 24 in., weighing nearly 1200 lb. The tractor and shovel left Winnipeg on July 4, were delivered July 11 and were in use July 15. Also flown to the mine were a 3-ton gasoline locomotive, twenty 24-cu. ft. mine cars, and railway equipment, the whole totaling around 200 tons.

The Airplane's Special Place in Transportation

At the present time, it appears that the airplane can serve the mining areas of Manitoba in special though perhaps limited ways. The plane is the most effective means of moving men and certain material. In prospecting, the season for which has been limited to the period between the middle of May and the middle of October, the introduction of aviation has lengthened that season and correspondingly increased the chances of mineral discovery. Formerly the prospector desirous of entering Manitoba's pre-Cambrian field was obliged to wait until the ice had cleared away from the northern water routes before leaving Winnipeg or The Pas. In the fall he would have to anticipate freeze-up by at least three weeks in remote areas. The actual time in the field would thus be shortened by at least six weeks. This time spent in travel constituted a serious loss in a prospecting season of 22 weeks. With aviation as an aid to the prospector, the airplane can land men in remote areas as soon as the ice breaks up in the spring or can pick them up a day or two before the lakes freeze in the fall.

Similarly, in moving survey parties the plane has brought about a great saving of time, and as it serves the prospector and

surveyor, it also serves the geologist in making a preliminary investigation of an area. Its value in picking out transport routes has been brought out forcibly where the scout desires to make a quick survey of the area through which the road is to pass. A further use of the airplane along Churchill River, in a survey for water-power development, proved that in this particular work both money and time could be saved.

For travel into areas where road building is difficult, the airplane will continue to be used, as it is the only practical means of reaching many places in the pre-Cambrian northland. On account of its speed, it has become a very important factor in opening up and developing the northern mineral resources. This service is so outstanding that today the airplane is a vital necessity in the development of the hinterland into which good summer roads can be built only at prohibitive cost and into which even over water routes access is slow if not difficult.

From its beginning as a carrier into the mining areas, the airplane has provided the most popular transportation for passengers and mail. It has not, however, become a low-cost freight carrier and unless these charges can be reduced to at least 50¢ a ton-mile other methods of transportation will continue to find favor in handling heavy material and large quantities of supplies. So far, the rates for freighting are nearly double this figure.

Nevertheless the airplane is exceedingly important as a carrier throughout Manitoba's pre-Cambrian areas. It is reckoned that of the freight carried in Canada by air 90 per cent is for the mining industry, and to this Manitoba has been a large contributor.

The value of the airplane to the mining industry in Manitoba is seen to greater advantage at the gold-producing mines, particularly when they are isolated. Compared with the base-metal mines, the gold mine has a much smaller product to ship out, and this can effectively be done by airplane. The first gold to be transported in that manner was from Central Manitoba Mines Limited in 1927. The base-metal mines must be served by railway both for incoming and outgoing freight, and unless a railway has other and more important interests to serve the smaller developments do not justify the building of a railway.

While the problem of communication with the railway continues to exist, the particular location and condition of a mine decides what method of transportation is to be used for the secondary hauling. At present in Manitoba the tractor performs a distinct service in winter freighting while the airplane offers an equally distinctive yet complementary service.

Rubber-tired Mine Haulage in the Tri-State District

By S. S. Clarke,* Member A.I.M.E

(New York Meeting, February 1942)

The sheet-ground deposits of the Tri-State district, because they are fairly uniform in thickness (7 to 11 ft.)—rather flat, with an easy dip to the west—and cover a large acreage, offered a problem of improved transportation.

Heading advance in this kind of mine is very rapid and track haulage requires a larger crew of trackmen than do other mines, and a heavy investment in track.

The use of some type of rubber-tired truck equipment for underground haulage was under consideration for some time by the Eagle-Picher Mining and Smelting Co. The nature of some ore bodies made them not adaptable to truck haulage and some mines that did offer possibilities were equipped with other types of haulage, and the expected savings by the use of trucks did not offset the loss of investment that would be incurred in discarding other equipment. The decision was made to sink a new shaft, designed for skip hoisting, on the Blue Goose ore body, and then develop truck equipment.

Developing Tractors and Trailers

A study was made of internal-combustion engines for power units in the tractors, but the extreme difficulty of controlling air circulation, and the heavy cost of drilling large churn drill holes for blower installations, led to the adoption of a tractor with a storage battery.

The storage-battery tractor that was on the market for mine haulage was impossible to obtain because all were needed for defense work. Several standard electric-truck chassis were purchased and rebuilt in the Eagle-Picher shops. The steering wheel was shortened and set vertical, permitting the driver's seat to be moved forward, and the frame was shortened enough to take the battery box and the fifth wheel support that carried the gooseneck of the dump trailer. The fifth wheel, or turntable, is of the ball and socket type. Fig. 1 shows the ball fastened to the gooseneck. The socket is split to permit introduction of the ball and is held by through-bolts, and the socket is welded to a ½-in. plate, which in turn is attached to the tractor frame. Smaller wheels and tires were put on to reduce over-all height. If it had been necessary, the springs could have been reversed, or taken off entirely, to gain additional clearance.

The tractors are equipped with 48-cell, 23-plate, 400-A.H. batteries. With the initial tonnage and mileage, the batteries were about 50 per cent down at the end of a shift.

Tractors have truck tires 7.50 by 20, 8-ply, front and rear, the rear being dual. After four months of use, the tires are beginning to show signs of failure.* In

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* General Superintendent of Mines, Eagle-Picher Mining and Smelting Co., Miami, Okla.

* It should be noted that this installation was made in September 1941, and the life of the original tires was comparatively short; with more experience in road maintenance and a new type of tire tread, tire life has been nearly tripled.

The tire-rationing board has recognized the zinc industry as essential, and so far no difficulty has been met in obtaining two or three retreads per month and a new casing every three or four months (September 1942).

future, tires will be special rock-tread, single instead of dual at the rear, as small sharp rocks wedge between the dual tires and chafe or cut the walls. The truck speed is not sufficient to throw out the rocks.

determine whether the springs are of any benefit. In fact, the tires on the spring-equipped trailer show more wear than those with solid axles. Single tires, 9.00 by 18, 10-ply, are used.

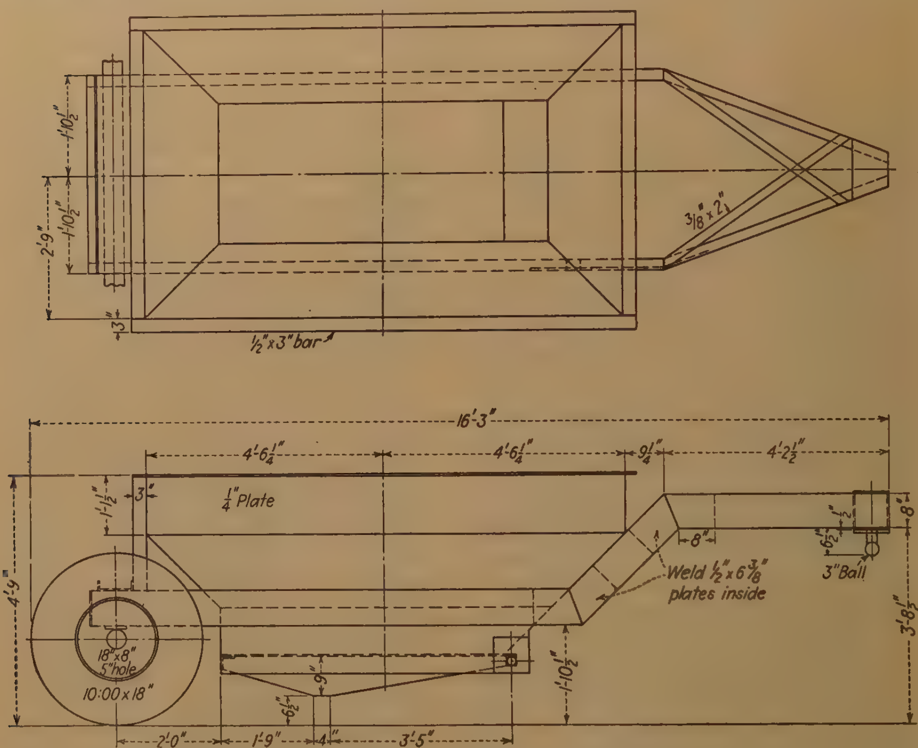

FIG. 1.—DETAILS OF TRAILER.

The trailers in use are small replicas of the larger trailer used on the surface to haul ore rock from outlying shafts to the central mill. They are simple in design (Fig. 1), easy to build, and maintenance costs are low. The capacity of the mine trailer, with a 2½-ton tractor, is 5 tons, and 7½ tons with the 5-ton tractor.

The trailer is built of ¼-in. plate, stiffened with ½ by 3-in. bands, ½ by 3-in. bottom angles and ⅜-in. rib plates, and mounted in an 8-in. channel frame. One trailer has two coil springs on the axle, while two have the axles U-bolted to the frame. So far, we have been unable to

The trailer weighs 2060 lb. and the tractor complete with battery weighs 9570 lb. The load distribution is about 40 per cent on tractor and 60 per cent on the trailer axle.

Trailers are equipped with a large bottom drop door, which trips as the trailer goes over the skip pocket and is latched by a closing device as the unit leaves the station. Fig. 2 shows the trailer over a skip pocket.

OPERATION

The equipment makes its own roads; only large rocks are raked off. Wheel traction is good. The tractors start slowly with

a load, without spinning. Some experiments are being made to improve the road, by mixing crushed limestone with the mine rock (chert), with the idea of getting a better bond of the loose material.

tors are capable of 7 to 8 miles per hour loaded, and 10 to 11 miles per hour empty. The trailers are loaded by a scraper loader, mounted on a caterpillar-propelled ramp (Fig. 3).*

Fig. 2.—Rubber-tired tractor and trailer. Trailer over skip pocket.

Fig. 3.—Trailer under scraper loader on ramp.

The three units in operation average 48 trips, totaling 240 dry tons each per shift and traveling about 650 ft. per round trip, or approximately 6 miles per day, at a speed of about 5 miles per hour. The trac-

The haulage units at present are not pushed to capacity, because of lack of

* For further details see S. S. Clarke: Development of Scraper Loading in the Tri-State District. A.I.M.E. *Tech. Pub.* 1115 (*Min. Tech.*, Sept. 1939).

minable headings. New development will open more headings, and we expect that the tonnage can be increased to an average of 350 tons per shift. The time study in Table 1 shows that these units are capable of handling a larger tonnage.

TABLE 1.—*Time Study*

Unit	Headings					
	No. 1 South		No. 2 North		No. 3 East	
	Time	Accum. Time	Time	Accum. Time	Time	Accum. Time
Loading time	3′-19″	3′-19″	1′-56″	1′-56″	2′-55″	2′-55″
Haulage time	1′-47″	5′-06″	2′-00″	3′-56″	2′-15″	5′-10″
Loading time	2′-44″	7′-50″	2′-36″	6′-32″	2′-10″	7′-20″
Haulage time	1′-44″	9′-34″	2′-59″	9′-31″	2′-30″	9′-50″
Total 2 trips, 10 tons		9′-34″		9′-31″		9′-50″
Round trip		550 ft.		650 ft.		750 ft.
Length of drag		90 ft.		65 ft.		65 ft.

TABLE 2.—*Haulage Costs, Comparable Sheet-ground Mines*

	Blue Goose 2 Mine	Mine 1	Mine 2	Mine 3	Mine 4
	Cost per Dry Ton				
Labor	$0.033	$0.087	$0.161	$0.168	$0.110
Repairs and supplies	0.009	0.027	0.043	0.024	0.018
Power	0.008	0.008	0.016	0.017	0.010
Total	$0.051	$0.129	$0.222	$0.211	$0.140

Number of Men and Cost per Dry Ton

	Men	Cost	Men	Cost	Men	Cost	Men	Cost	Men	Cost
Track men			2	$12.60	2	$12.60	2	$12.60	1	$ 6.30
Hoist men			2	13.10	3	19.65	2	13.10	2	13.10
Rope rider			2	12.10	3	18.15	2	12.10	2	12.10
Maintenance men	1	$6.30			2	12.10	2	11.10	1	5.55
Truck drivers	3	18.90								
Total	4	$25.20	6	$37.80	10	$62.50	8	$48.90	6	$37.05

EFFICIENCY OF PERSONNEL

	Blue Goose 2 Mine	Mine 1	Mine 2	Mine 3	Mine 4
Tons per man on haulage	181.0	68.5	40.4	46.2	60.1
Tons per total men employed	13.9	10.9	9.7	8.3	11.7

COST OF HAULAGE EQUIPMENT

Mine	Equipment	Cost
Blue Goose 2	Tractors, trailers and charging	$ 9,550.00
Mine No. 1	Trackage, cans, cars, hoists, cables, etc.	12,640.00
Mine No. 2	Trackage, cans, cars, hoists, cables, etc.	16,500.00
Mine No. 3	Trackage, cans, cars, hoists, cables, etc.	15,650.00
Mine No. 4	Trackage, cans, cars, hoists, cables, etc.	8,100.00

The Blue Goose No. 2 mine is a typical sheet-ground mine, having a roof height of about 11 ft. Drilling and blasting are done on night shift, loading and hoisting on day shift. One scraper loader and one haulage unit, as a rule, work together. As length of

haul increases, it may be necessary to put another haulage unit in service to avoid any loss of output.

Four other mines are in the sheet-ground horizon and conditions are comparable with Blue Goose No. 2. Mechanical loading is used and the typical 32 by 32-in. cans and ground cars are the haulage equipment. Two-drum or four-drum tail-rope haulage hoists are used for spotting the cans under the loader ramp, as well as to pull the train to the shaft.

Costs

Table 2 shows a comparison of operating costs and efficiency of personnel of comparable sheet-ground mines.

Conclusion

The use of rubber-tired haulage equipment has been so successful in its flexibility, lower costs, and reduced investment charges, that it offers many possibilities, especially in reopening old mines from which all tracks have been removed.

Operation of Diesel Locomotives Underground

By Fred W. Stiefel,* Member A.I.M.E.

(New York Meeting, February 1942)

This paper covers the operation and maintenance of Diesel locomotives underground on a portion of the Delaware River Aqueduct.† This part of the tunnel is 15 miles long, with shafts 14 ft. in diameter spaced about 5 miles apart. These shafts (2, 2*a* and 3) are 825 ft., 1551 and 840 ft. deep, respectively. The grade of the tunnel is almost level, except for the 12,100 ft. north of shaft 2, which has a grade of 2.31 per cent. The tunnel is circular, 17 to 19 ft. in diameter. It lies at a depth of 400 ft. below sea level, with a maximum overburden of 2400 feet.

The tunnel driving was carried on in six headings, using 15 storage-battery locomotives, each of 13 gross tons and having a maximum speed of 8 miles per hour. The longest haul was 3 miles, which proved to be the maximum distance for which storage-battery locomotives could be employed economically for the amount of material to be moved. One locomotive was used for switching cars at each heading, one for switching cars at the bottom of each shaft and one on the main-line haul from the heading to the shaft.

The maximum amount of material hoisted in one shaft from two headings when 104 ft. of tunnel was driven in 24 hr. was 1000 cu. yd., or about 2000 tons of rock in place. The total quantity excavated was 750,000 cu. yd. (1,500,000 tons of rock).

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* Chief Engineer and Manager, Samuel R. Rosoff, Ltd., Kerhonkson, N. Y.

† Tunnel Contract No. 313, Board of Water Supply of the City of New York (Samuel R. Rosoff, Ltd., contractor).

The tunnel-lining operations using concrete presented a problem of hauling a maximum distance of 5 miles in the tunnel. The concrete-placing equipment had a capacity of about 2000 cu. yd., or 4000 tons in 24 hours.

Experience had shown that storage-battery equipment could not handle this quantity of material within the time required at the distance of 5 miles. Trolley-locomotive haulage was considered dangerous and expensive, and would cause interference because transmission equipment would be required. Therefore the performance of Diesel locomotives of 160 hp., with a speed of 18 miles per hour, in comparison with that of other types of locomotives, was studied.

Choice of Locomotives

In deciding upon the type of locomotive to be used for the 5-mile haul, four alternatives were considered, as follows:

1. To purchase additional storage-battery locomotives with the necessary batteries over the 6 locomotives and 12 batteries available.
2. To convert the existing battery locomotives to the trolley type.
3. To purchase new trolley locomotives.
4. To purchase Diesel locomotives.

Proposition No. 1

The 90-hp. storage-battery locomotives available (each weighing about 12 tons), when hauling a load of 64 tons, with a drawbar pull of about 1900 lb. on a level track, developed a speed of 8 miles per hour and could make five round trips in 8 hr. with a

power consumption of approximately 46 kw-hr. Since the capacity of the 29 plate-54 cell battery was 81.32 kw-hr., on the 5-mile haul a battery would require charging after each round trip. It was estimated that no more than six trains could be efficiently and conveniently operated. This would have required the purchase of 18 additional sets of batteries and boxes, and and six additional charging sets, at a total cost of over $100,000. It was necessary to charge 30 batteries per shift. Space in the tunnel would have to be provided to accommodate extra battery-charging equipment, with additional maintenance and operating labor. This eliminated the prospects of the use of storage-battery locomotives.

Proposition No. 2

The cost of converting the storage-battery locomotives to the trolley type, the installation of trolley and transmission facilities and the required booster stations would have cost $70,000, not including the additional maintenance and operating cost of this system.

Proposition No. 3

Proposition No. 3 is similar to Proposition No. 2. It contemplated the purchase of six new trolley locomotives with trolley and transmission equipment, three new 150 kw. motor-generator sets and, not considering the maintenance and operating expense, would have cost $100,000.

Proposition No. 4

With a Diesel locomotive it is possible to make seven round trips at a distance of 5 miles in an 8-hr. shift, with a capacity of 87½ cu. yd., or 175 tons per hour. This would require five locomotives, costing $58,500.

Comparison of Costs

The cost price of six new storage-battery locomotives with battery boxes and charging sets would be $188,608 (as compared with $58,500 to purchase five Diesels to do the same work).

Table 1.—*Comparison of Estimated Operating Costs*

Per Day

Item	Each	Total
Six Electric Locomotives		
18 motormen	$13.20	$237.60
18 brakemen	7.00	126.00
6 electricians	13.20	79.20
Insurance at 15 per cent		66.42
Total		$509.22
Five Diesel Locomotives		
15 motormen	$13.20	$198.00
15 brakemen	7.00	105.00
4 maintenance men	14.17	56.68
Insurance at 15 per cent		53.95
Total		$413.63

A comparison of the estimated operating costs for the maximum haul for the electric and the Diesel locomotives is given in Table 1,

The minimum haulage requirements at the shortest haul is two locomotives per shift. The number of maintenance men and electricians remains the same, therefore the average daily cost is as follows: electric locomotives, $369.84 per day; Diesel locomotives, $309.10 per day.

The comparison in total labor cost for the electric and Diesel haulage for the complete job was:

Electric locomotives	350 × $369.84	=	$129,444
Diesel locomotives	300 × 309.10	=	$ 92,730

Hence, the saving in labor and insurance alone by using the Diesels would be $36,714 for 400,000 cu. yd. of concrete.

Actual Diesel operating costs for a period of one month is as follows:

Labor cost	$7,609.00
Insurance at 15 per cent	1,141.35
Fuel and lubrication	429.00
Repairs	600.00
Total cost per month	$9,779.35

The total mileage covered was 11,017 miles. Each train hauled 20 cu. yd. of concrete, or a total of 40 tons. The cost for the month was $0.0222 per ton-mile, covering all expenses.

Ventilation

In response to a request by Daniel Harrington, Chief of the Health and Safety Branch of the U. S. Bureau of Mines, to test Diesel locomotives underground under propitious circumstances and "to be frank in pointing out not only the good features, but also any of the less desirable ones for the Diesel system," I wish to state that considerably more ventilation is required in the operation of the Diesel locomotives underground than is required in the use of the storage-battery or trolley locomotives.[1] More careful operation and more scientific maintenance is required to keep them in proper operating condition. Some people may consider these features a disadvantage, but generally I believe them to be an asset to mining. Copious ventilation in mining operations has been recognized as necessary to cope with dust and gaseous conditions, to obtain better visibility, to enhance safety and to speed up the work. If the Diesel locomotive establishes no other beneficial advancement, it positively promotes liberal ventilation.

Atmospheric conditions where Diesel engines were used in tunnels are discussed by Ash and Naus.[2]

It was found that 10,000 cu. ft. per min. of ventilation was required for each Diesel locomotive, in order to avoid objectionable gas concentrations. Daily observations were made of top and tunnel temperatures, barometric readings, and ventilation, so that proper control over atmospheric conditions was maintained. Except for high temperatures produced by the concrete, together with a humidity of 80 to 90 per cent, no inconvenience was suffered by the tunnel workers. The highest surface temperature experienced upon the work was 100° above zero and the lowest was 17° below. The general tunnel atmospheric temperature throughout the year was 70°.

In about two hours without ventilation, an explosive concentration of methane gas could develop in the tunnel to the north and south of shaft *2A*. The ventilation required by the Diesel locomotives obviated any such danger.

Design of Locomotive

It appears that the Diesel locomotive can take the place of other locomotives in underground main-line haulage where plentiful ventilation can be provided economically and reasonably, or where generous ventilation is adopted because of other controlling conditions. Greater speed is safely obtained by the Diesel locomotives over other types because of its low center of gravity, which reduces chance of derailment. On a well-constructed road bed, with properly constructed cars, it is safe to operate a train at 20 miles per hour. Heavier rails than for the slower locomotives are required, and under similar operation to that herein described no less than 80-lb. rails should be used, with the proper spacing of ties and well-constructed joints.

As much as 3000 to 3400 tons of concrete have been placed in the concrete arch lining in 24 hr., at a 5-mile haul. During mucking operations a single Diesel locomotive hauled a train of 20 muck cars of 6 cu. yd. capacity, each car having a gross load of 11.6 tons. The total trainload moved (including the weight of the locomotive) was 245 tons on a level track.

The Diesel locomotive weighs 14 tons and is 6 ft. 2 in. high over all. Gauge is 36 in.; drivers dimension, 28 in.; weight on drivers, 28,000 lb.; length over all, 15 ft.

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[1] The safety features have been covered in a paper by W. B. Harris, L. Greenburg and G. Werner (p. 165, this volume).

[2] S. H. Ash and L. L. Naus: Use of Diesel Locomotives in Tunnels. Bur. Mines *Inf. Circ.*

9½ in.; wheel base, 4 ft. 9½ in. (wheels inside frame).

The basic design of these locomotives consists of a four-cycle, six-cylinder, 5-in. bore × 6-in. stroke Diesel engine. To this engine is mounted a conventional type four-speed transmission and clutch unit, which in turn is connected through a universal joint to an axle-hung reverse gear transmission, mounted on the rear axle and permitting four speeds in either direction of travel. The front axle is driven from the rear axle through sprockets and heavy roller chain. The locomotives are of the 0-4-0 type, having driving wheels 28 in. in diameter, axle bearings of heavy-duty roller type mounted in cast steel journal boxes, supported by semielliptic springs, cross equalization giving maximum riding qualities over rough tracks. Side frames and bumpers are of heavy rolled steel slab, welded, with all necessary brackets and supports welded in place. An efficient hand brake of the lever type, actuated by heavy screw and hand wheel, is at the operator's position. Brake mechanism can be arranged for a straight air brake, if desired. The superstructure consists of a heavy steel-plate removable hood, having necessary hinged inspection doors and sloping sides, which permit maximum visibility from the operator's station at the rear end of the locomotive. Couplers, steps, hand holds, guard rails, warning signal and operator's seat are provided.

The flameproof, exhaust scrubbing and safety features of this mine locomotive consist of the following units: The main fuel-injection pump of the Diesel engine is set and permanently sealed for a ratio of air to fuel of not less that 20 to 1, to assure maximum combustion of fuel and minimum CO_2 content of exhaust gases.

Fuel for the injection pump is supplied by the fuel-transfer pump on the engine, from a supply tank situated between frames at the front end of the locomotive. The fuel system is provided with proper filters and the fuel tank has a Protectoseal fuel-filler cap. The fuel line is equipped with a safety shutoff valve, which can be tripped from the operator's station in an emergency.

The air for the engine intake is taken through a double-unit oil-bath air filter mounted on the manifold labyrinth, which in turn is mounted directly to the engine air-intake manifold. The function of this air-intake labyrinth is to arrest flames caused by backfiring of the engine. It consists of a heavy, cast flameproof chamber, equipped with cleanout and inspection covers, in which is mounted a unit made up of thin brass disks spaced to form a labyrinth. This unit is easily removed for cleaning purposes should any carbon deposits occur.

The engine exhaust is carried from the engine through a water-cooled manifold to the scrubbing and cooling tanks, through heavy steel piping, and from the scrubbing tanks to the exhaust labyrinth (through steel piping) and thence to the fan chamber, where it is mixed with air drawn through the radiators of the cooling system of the engine. The diluted exhaust finally is discharged through a rectangular opening in the side of the locomotive hood opposite the operator's position.

The purpose of this exhaust system is to quench any possible sparks or flames and to cool, dilute and clean the exhaust of the engine.

Mechanical Troubles

In the early operation of the Diesel locomotives, some serious mechanical trouble developed, attributable partly to improper operation and maintenance. The manufacturers, swamped with defense orders, had left us to develop our own experience. Had we obtained the benefit of their service, considerable difficulty would have been avoided. Pistons seized in two locomotives and the piston rods broke through the walls of the engines. These failures were due either to overspeeding the engine,

poor injection of fuel oil or the breakdown of lubricating oil, or the combination of all of these conditions.

After the two engine failures, analysis was made of lubricating oils as well as of fuel oil. Samples of used crankcase oil were analyzed. Our experience tends to indicate that the highly paraffinic types of oils appear to have superior load-carrying ability and present greater resistance to cylinder, piston and ring scuffing or abrasion. Unlike the fuel of the gasoline engine, where the gasoline is boiled out of the crankcase oil by high temperatures, the Diesel fuel oil finding its way to the crankcase will increasingly dilute the lubricating oil. Also, oil thickening results from the presence of suspended soot caused by incomplete combustion and oil oxidation. It was found that another contaminating material that tends to thicken the oil is moisture. Water in the fuel or lubricating oil can rapidly be destructive to a Diesel engine.

Poor mechanical operating conditions of the Diesel engine would create carbon monoxide, oxides of nitrogen, sulphur dioxide, aldehydes and smoke. This condition soon becomes evident and unless corrected causes engine breakdown.

Regulations Governing Operation and Maintenance

Each locomotive operator is compelled to make a daily report on a form issued by the company. They are ordered not to abuse the hydraulic coupling by idling in gear, nor to accelerate in high gear under loads. They are required to guard against excessive engine speed and to regularly observe temperature gauges and viscometer.

The maintenance men are required to change the lubricating oils regularly, observe the proper functioning of a double set of fuel-oil filters and check the nozzles frequently with a nozzle tester. Frequent compression tests are made of each engine cylinder. Correct timing of the fuel pump must be maintained. Bolts or other fastenings and electrical connections are checked daily. Fan belts are gone over weekly. In rotation, locomotive engines are thoroughly overhauled about every two or three weeks, so that each engine is so taken care of every three months.

In the following paragraphs are listed the maintenance regulations that were issued to each operator and maintenance man.

Maintenance Instructions for Whitcomb Diesel Locomotive

This locomotive has a Hercules Diesel engine, Bosch fuel injection equipment and American Blower Company hydraulic coupling which need particular care and maintenance in order to keep them in economical operation and to guard against breakdown:

1. *Engine Knocks.*—If the engine knocks or shows any disturbance, the cause must be found and corrected immediately.

Note instructions to Diesel locomotive operators, a copy of which is hereto attached.

2. *Fuel Injection.*—The fuel injection, which is controlled by fuel pump timing and by the nozzle pressure setting, with proper governor control, must be properly synchronized to prevent engine trouble.

3. *Injection Nozzle.*—The nozzle must operate with a proper spray into the precombustion chamber, to prevent carbon from clogging the nozzle and development of gumming of fuel, which may result in piston seizing due to "frozen" piston rings.

Clean spray nozzles: (1) When engine exhaust shows black or dark smoke; (2) in loss of power with foul exhaust; (3) when engine runs ragged; (4) irregular fuel knocks. Thus, nozzles must be tested regularly and fuel-pump pressures noted.

4. *Lubricating-oil Pump and Oiling System.*—Oil filters should be inspected daily and cleaned of dirt.

Observe oil pressure: Gradual lowering of oil pressure indicates cleaning of filters may be necessary.

With oil hot, correct oil pressure is about 30 lb., engines running full speed. At *No-Load* speed, when idling at 450 r.p.m., the pressure should be about 8 pounds.

Cleaning of filters requires knowledge of the procedure; after cleaning replace gallon of fuel oil by using vent valve; shut valve tight.

Clean oil filter when oil is changed. Air must be out of filters, add extra gallon of oil.

If after cleaning filters a higher pressure is shown than under former operation, it indicates filters were not cleaned often enough.

Clean filter carefully. *Do not use wire brushes, or force. Clean with air, soft brushes or rags only.* Replace gaskets to avoid oil leaks. Check oil level in injection pump base daily. Drain filter daily. Check fuel-pump drive weekly.

5. *Viscometer.*—The whole instrument needs cleaning periodically. Filter screen should be cleaned every time the oil is changed. Use only compressed air to clean resistance tube.

A thin oil is indicated by low pressure; a thick oil by high pressure. If oil shows low viscosity, engine must be stopped and oil changed immediately.

6. *Water Pump and Water-circulating System.*—The cooling water must at all times be clean and acid free. The water passes through relatively small holes. Never operate with water boiling. Add water daily. Grease pump bearings daily. Observe if scale or sediment is forming. If water pump leaks tighten packing nuts or renew packing.

Inspect radiator weekly; clean if clogged or shows scale; report scaling to Master Mechanic.

Blow with air weekly to clean dust and dirt from fins and outside of tubes. If scale is found inside of radiator this must be removed with some approved solvent such as Young cooling system cleaner.

7. Observe the Viscometer, Oil pressure and ammeter often. *Check oil level daily.*

8. *Fuel-injection Pump.*—This pump has the same function as the ignition system of a gasoline engine, therefore must be properly timed.

The fuel injection must take place at maximum air compression of the engine.

Do not attempt to repair this pump. It must be sent to manufacturer for overhauling. Spare pump is on hand.

Permit no engine to operate when improperly timed.

The pump must be timed at the exact moment of closing—not just before or after. See Hercules Instruction Book for timing fuel injection pump, pages 35 to 39.

When new pump is put in service, or after overhauling, the pump base is empty and must be filled. Use gauge rod opening to fill.

Check oil level in injection pump each day and keep oil level mark on gauge rod. Use same oil as supplied to the crankcase.

The strainer should be cleaned often. If clogged, engine will not operate properly. Clean carefully and do not puncture a hole in strainer.

Lubricate oil cups daily with same oil used in fuel injection pump and governor lubrication.

9. *Fuel-oil Strainers or Filters.*—These strainers must be kept clean in order to keep pump plungers and fuel nozzles clean, to prevent dirt from reaching pump and fuel nozzles. Wash filters in clean fuel oil.

There should be three separate filters on the fuel-oil line, each kept clean. Dirt in fuel oil rapidly damages injection pump. Drain filter daily. Remove air from fuel filters by opening vent cocks weekly. Hand-pump until oil runs out freely; close while still pumping.

10. *Fuel Lines.*—Keep fuel lines in stock. Longest line will fit shortest or any other. *Do not braze or solder line.*

11. *Fuel Nozzles.*—The accuracy of the manufacture of the fuel nozzles requires clean fuel oil. Keep nozzles absolutely clean at all times.

12. *Governor.*—Governor should maintain engine speed under "no load" of less than 1750 r.p.m. or less than 1600 r.p.m. with full load speed.

Governor must be kept sealed at all times and adjusted only by maintenance man, and records made of same.

About ½ teacupful of oil should be added every second day, using same oil as used in crankcase. If oil is drained from governor, ¼ of a pint of oil should be replaced.

Remove oil-level plug in housing and check oil. Fill oil level with top of hole.

13. *Air Cleaners.*—Air cleaners not properly maintained may result in badly worn cylinders in a few days. Clean daily if necessary; more often in dusty atmosphere.

14. *Fuel Oil.*—Each delivery of fuel oil should be tested by the "burn-out" test.

Oil must be clean, with best burning characteristics. Clean filters often.

Oil must not contain water. Drain fuel supylp tank weekly and wash out with clean fuel oil to remove all dirt or sediment.

15. *General Notes.*—If engine has not been operated for week or more, remove nozzle holders and by means of spout can inject 2 to 3 tablespoonfuls of lubricating oil. Water on top of cylinder may wreck engine. Turn over by hand with nozzle holder out to relieve compression.

Remove head every three months to examine piston rings.

Inspect fan belt; adjust once a week.

Dirt is the greatest enemy of the engine; maintain all filters constantly.

Use only clean water for cooling system.

There should be no sign of smoke from the exhaust. White, light blue, dark blue, brown or black smoke is a result of improper combustion.

Tighten all loose bolts, nuts, screws and electrical connections daily.

Check battery water daily.

Tighten cylinder-head nuts and inspect valve clearance weekly.

16. *Exhaust Labyrinth.*—The exhaust labyrinth should be examined every month and cleaned. If filter material is carboned badly, it must be cleaned or replaced. To test this labyrinth remove cover and pour water in quickly from a large bucket; if water runs through rapidly the labyrinth is O.K. Exhaust water boxes must be drained and flushed every two weeks, otherwise there may be a bad odor in the exhaust.

Diesel Engines in Tunneling Operations

By William B. Harris,* Leonard Greenburg* and Gustav Werner†

(New York Meeting, February 1942)

Haulage in tunneling operations generally has been done with electric locomotives. As a rule, on short hauls the source of electricity is a storage battery mounted on the locomotive, which, of course, must be removed and recharged periodically. Where the distance from the recharging point to the tunnel face is very long, and where loads are heavy, trolley-driven locomotives are used. Diesel engines have been used in mining and tunneling operations in Europe for a number of years. The recent reports of studies in Belgium[1] and by the Bureau of Mines in this country[2-4] have indicated the potentialities of this form of power for haulage, mucking, bulldozing and many other underground operations.

The hazards attending the use of Diesel engines in more or less closed spaces have long been known. The exhaust constituents that may create a harmful or objectionable environment are carbon monoxide, oxides of nitrogen and sulphur, carbon dioxide, aldehydes and smoke. Lack of precise information regarding the concentration of these gases and the ventilation requirements to reduce them to safe levels has to some extent suppressed their general acceptance in tunneling and mining work. Not until the work of the Bureau of Mines referred to above, and certain experience gained in the construction of the Delaware Aqueduct,‡ has there been available the requisite information regarding the conditions under which their use may be permitted. This paper concerns itself with the successful operation of Diesel engines underground in the construction of certain parts of the Delaware Aqueduct, and describes the conditions under which Diesel engines have been found to operate with a minimum production of noxious gases. The test data reported were obtained during the operation of Diesel engines over a period of two years.*

Studies by the New York State Department of Labor

Exhaustive tests were made by the Division of Industrial Hygiene, of the composition of the exhaust gases given off by a Diesel engine mounted on a test block. These tests showed that certain quantities of the gases are discharged from the exhaust. On the basis of these figures, preliminary requirements of design and control

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* New York State Department of Labor, Division of Industrial Hygiene, New York, N. Y.

† New York State Department of Labor, Bureau of Mines, Division of Inspection, New York, N. Y.

[1] References are at the end of the paper.

‡ The Delaware Aqueduct is a continuous hard-rock tunnel, construction of which was begun in 1937. This circular tunnel, averaging about 17 ft. in diameter and 85 miles long, when put into use will add about half a billion gallons of water daily to the New York City water supply. The use of Diesel locomotion in this work meant a great saving in time and cost to the contractors, as some of the haulage lines were as long as 35,000 feet.

* Recommendations leading to the use of Diesel engines underground were prepared by the Division of Industrial Hygiene, New York State Department of Labor, in conjunction with the Division of Mines and Tunnels of the same Bureau, the New York City Bureau of Water Supply and the U. S. Bureau of Mines. The recommendations followed upon the application of one of the Delaware Aqueduct contractors, who agreed to certain stipulations regarding the controls described later in this paper.

were drawn up and the use of the engines in the tunnel was permitted.

After the Diesel engine was put in operation underground, samples were taken of the air in the tunnel and determinations were made of the gases present in the tunnel atmosphere. It was found that the oxides of sulphur could be maintained at a low level if the sulphur content of the engine fuel was less than 0.5 per cent. Although small quantities of oxides of nitrogen were present in the air, they were found essentially to vary directly with the carbon monoxide content. The same was true of aldehydes. From a study of the tunnel-air samples, the amount of carbon monoxide present was found to be a direct measure of the toxicity and the irritating properties of the atmosphere. In order to evaluate the effectiveness of the ventilation and the gas production of the Diesel engine, therefore, samples were taken of the air to determine its carbon monoxide content. The content of this gas was found to indicate fairly accurately the degree of control afforded.

The tunnel into which the Diesel engine was introduced was ventilated by a system of direct blowing. In spite of the fact that the recommended quantity of ventilation was being used, it was impossible to reduce the air contamination throughout the tunnel to a safe concentration. With the cooperation of the contractors engaged in the construction of the Aqueduct, a completely different method of ventilation was installed on this tunnel. The type of ventilation used will be discussed in another part of this paper. Studies carried on after this new system was installed showed that the recommendations set forth by the Department were somewhat inadequate and a new set of requirements was drafted.

Design and Operation

On the basis of preliminary studies, the New York State Department of Labor approved the use of Diesel engines in the Aqueduct subject to the conditions outlined in the requirements listed below. These requirements have been quoted by D. Harrington and S. H. Ash,[6] of the United States Bureau of Mines, who have cooperated in this work.

Design and Maintenance

1. The engine fuel shall be equivalent in quality to that specified by the engine manufacturer and shall have a sulphur content not exceeding 0.5 per cent. It shall be kept in clean condition and shall be free from all foreign contaminants before being used in the engine.

2. No fuel shall at any time be stored underground except that carried in the fuel tanks of the engine.

3. The fuel injection system shall be designed and constructed so that the ratio of air to fuel cannot be reduced below 20 to 1, and the fuel pump adjustment shall be sealed.

4. An extra fuel pump set to meet the above requirements and sealed shall be furnished for each engine.

5. The engine manufacturer shall specify: (*a*) minimum idling engine speed, (*b*) maximum engine speed, (*c*) maximum power output.

6. The engine shall be equipped with flame arrestors on the intake and exhaust systems, of such a design that the surface temperature of the exhaust manifold shall not exceed 400°F. and the exhaust gas temperature at the point of discharge shall not exceed 160°F.

7. The engine exhaust gases shall be discharged into the atmosphere at a point remote from the engine operator.

8. The exhaust gases shall be cooled either by washing and/or spraying with water so that no incandescent particles shall be discharged into the surrounding atmosphere.

9. Spent gases from the Diesel engine shall be diluted with at least 10 times their volume of air before being released into the underground atmosphere.

10. The level of water in the storage tank shall control a valve interlocked to the fuel line which will stop the flow of fuel oil before the water supply becomes exhausted.

11. The engine shall be equipped with an electrical or other starting mechanism of sufficient capacity to permit frequent starting of the engine.

12. The carbon monoxide content of the exhaust gases leaving the engine manifold and before dilution shall not exceed 0.1 per cent.

Toxic Concentrations

1. The carbon monoxide concentration at any point in the general tunnel air shall not exceed 0.002 per cent.

2. The concentration of oxides of nitrogen at any point in the general tunnel air shall not exceed 40 parts per million.

3. The concentration of sulphur dioxide, aldehydes, visible smoke or other irritants in exhaust gases shall not create toxic or irritating atmospheric conditions or reduce visibility in the tunnel.

4. Hazardous or irritating atmospheric conditions created by the engine but not covered by the above specifications shall be brought under control as required by standards of Industrial Hygiene.

Testing

1. The engine, including the auxiliary exhaust-gas conditioning equipment, shall be tested before being taken underground and also after being placed in operation, under the conditions for which approval by the Department of Labor has been requested for a period long enough to determine its operating characteristics.

2. The engine shall be subject to weekly inspection by a technical representative of the manufacturer, who shall certify that it is being maintained in proper working condition. Such inspections shall be continued during the period of test by the Department of Labor and thereafter shall be at periods determined upon by the results of the tests.

3. No locomotive shall be used in tunnel operations which may contain explosive gas unless classed as "permissible" by the United States Bureau of Mines.

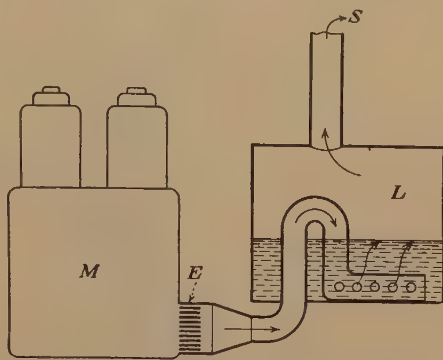

FIG. 1.—ARRANGEMENT WITH WATER DABBLING DEVICE.

M, motor; *E*, exhaust-plate device placed ahead of gas-washing circuit; *L*, gas-washing box (scrubber); *S*, gas issue.

Ventilation Requirements

1. Before operating a Diesel engine underground primary exhaust ventilation at a rate of 10,000 cu. ft. per min. or more per engine and a minimum air velocity of 30 linear ft. per min. in the tunnel with the end of the vent pipe not more than 150 ft. from the face nor less than 25 ft. beyond the car changer shall be maintained. The main vent pipe opening shall be at the crown of the bore.

2. Auxiliary ventilation shall be by blowing at a rate of 3000 cu. ft. per min. or more, with the intake to this system not less than 250 ft. outby the car changer and the discharge not more than 65 ft. from the face. The minimum air velocity between the face and the exhaust main shall be not less than 10 linear ft. per min. The location of the auxiliary vent pipe shall be at the "spring line." The auxiliary vent pipe shall be metal from the air intake to the car changer; flexible pressure tubing may be used between the car changer and the face. The intake to the auxiliary system must be so located that air entering it will not be

contaminated by exhaust gases from the Diesel locomotive when operating between the auxiliary intake and the face of the tunnel.

3. Whenever a Diesel engine is used in underground tunnel operation, all headings shall be separately ventilated by primary exhaust ventilation. When any portion of the primary exhaust ventilation system is not functioning, no Diesel motor shall be running.

4. The auxiliary ventilating system shall be put into operation as outlined above before the men return to work at the face after blasting.

Protective Devices Used on Diesel-engine Exhaust

The successful operation of Diesel engines underground is contingent upon the design of control apparatus connected to the exhaust manifold. Two considerations are of greatest importance with regard to the kind of apparatus used; namely, (1) the elimination of flame, and (2) the reduction of exhaust contaminants to safe limits. The first of these is a prerequisite in gassy mines and tunnels and for soft-coal mining operations, while the second is necessary in all types of underground operations. The devices used on Diesels with which this study is concerned are modifications of those developed in Europe.[1] Figs. 1 and 2 illustrate the basic principles involved. Flame elimination and partial cooling of exhaust gases is achieved by a plate device *E* shown in Fig. 1.

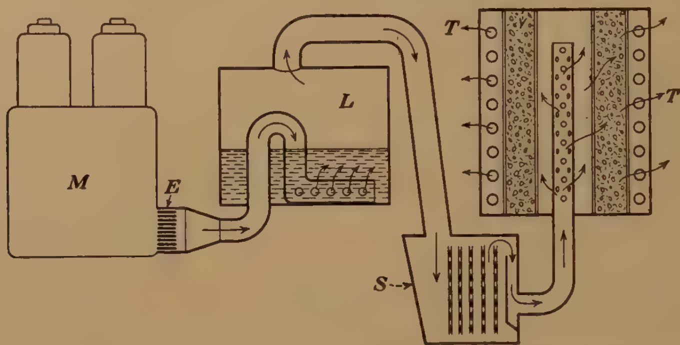

Fig. 2.—Arrangement including washing, drying, and circulating of gases through activated coal.

M, motor; *E*, exhaust-plate device placed ahead of gas-washing circuit; *L*, gas-washing box (scrubber); *S*, drying apparatus provided with baffle plates; *C*, activated coal; *T*, gas-exhaust ports.

After the gases pass through the flame eliminator they are forced through a labyrinth scrubber assembly, indicated by *L* in Fig. 2, which is partly filled with water. As the gases pass through the scrubber, most of the noxious contaminants other than carbon monoxide are removed.

From the scrubber, the gases pass through another labyrinth (*S* in Fig. 2) where the moisture picked up in the scrubber is removed. Thereafter, the gases are exhausted through a muffler. The parts indicated by *T* and *C* in Fig. 2, which is essentially an activated-charcoal absorber, were not used in the Aqueduct engines.

The general arrangement of exhaust-control equipment used on the Delaware Aqueduct engines differs in some essential respects from that shown in Figs. 1 and 2. As has been pointed out in the section on Design and Operation, three important requirements are set forth: (1) control of air-fuel ratio; (2) reduction of noxious gases and (3) dilution of final exhaust discharge. The first of these was achieved by means of

a suitable air-fuel valve, so that the air-fuel ratio did not fall below 20 to 1 (approximately equivalent to 65 cu. ft. per min. per brake horsepower). After the valve was set

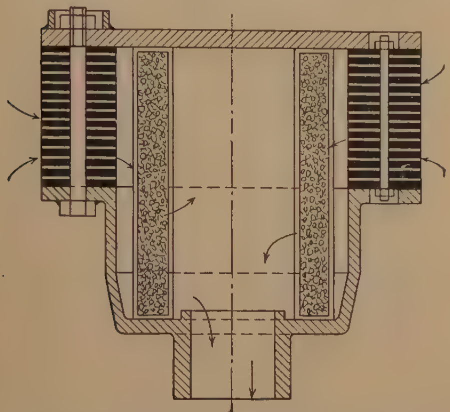

FIG. 3.—ANNULAR PLATE DEVICE AT INTAKE, SURROUNDING ANNULAR FILTER CONTAINING COPPER CUTTINGS OR SMALL CASINGS.

it was sealed, and the arrangement was such that when the air-fuel ratio fell below the value given the engine would automatically be shut off. The second and third requirements were achieved in the following manner:

The exhaust gas leaving the engine passed through a water-jacketed manifold to an exhaust cooling box. A water spray was tapped into the box and supplied from a special tank of water by means of a pump. The water from the spray was made to travel downward through the pipe in contact with the exhaust gas leaving the exhaust cooling box. The exhaust gas was discharged into the water contained in the cooling box through the perforated pipe. The gas then passed to the top of the box, where a perforated plate and a baffle plate were interposed to stop the spray. The gas was forced into a second cooling box designed to separate the water spray from the exhaust gas by means of another series of baffle plates. After leaving the second cooling box, the gases passed into a labyrinth of berles saddle. This tended to remove the soot and oil present in the exhaust gas. The discharge from the second cooling box was relatively free of noxious contaminants and was delivered to the fan behind the radiator, where it was diluted before it passed into the general air.

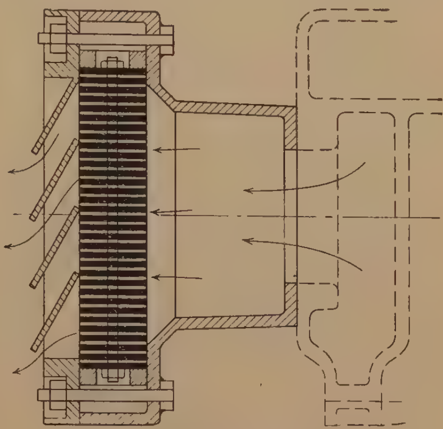

FIG. 4.—EXHAUST-PLATE DEVICE. Rectangular plates protected by means of blinds against outside percussion.

EXPERIMENTAL RESULTS

Methods of Test.—After preliminary testing showed that the quantities of aldehydes, oxides of nitrogen and other noxious materials were essentially contingent on the carbon monoxide content, testing for air contamination by the Diesel engines resolved itself to sampling for this gas. The instrument employed in the field for this was the widely used Hopcalite carbon monoxide indicator. This device as employed by the Department was fitted with a supersensitive scale in addition to the normal scale. The smallest division on this scale represents an atmospheric concentration of 5 parts per million (0.0005 per cent) of carbon monoxide. Samples of atmosphere were taken at three different locations: (1) in the exhaust manifold of the engine, (2) at the final exhaust of the engine (after dilution) and (3) in the tunnel air stream.

TABLE 1.—*Results of Tests on Diesel Locomotive at Factory*

Condition of Test	Engine Speed, R.P.M.	Carbon Monoxide[a] Concentration in Exhaust, Per Cent
Idling	530	0.00
Intermediate	700	0.01
High	900	0.01
High	1,700	0.01
High	1,600	0.01

[a] These samples were analyzed by the iodine pentoxide method.

Diesel Locomotives.—In cooperation with the United States Bureau of Mines, a series of tests was made on the exhaust of the Diesel locomotive. The results of these tests are shown in Table 1, and indicate that before final dilution with air the carbon monoxide contained in the exhaust gases is relatively low. These tests were made before any Diesels were used underground. Subsequently, during tunneling operations, six similar locomotives were tested underground, with results as shown in Table 2. The tests were made under widely varying conditions of tunnel ventilation but in general show less than the accepted standard for safe concentrations of carbon monoxide in air (100 p.p.m. = 0.01 per cent). Complaints from the tunnel workers were infrequent, except when locomotives began to smoke. On one investigation of such a condition, a distinct odor of exhaust gases could be detected in the tunnel air. No Diesel should be permitted underground when it smokes or is in need of repairs.

Tests on Diesel Trucks.—The Diesel engines were tested on nine trucks having capacities of 8 to 15 tons. Four of these

TABLE 2.—*Results of Tests on Diesel Locomotives in Tunnel*

Rated horsepower (1600 r.p.m.)	160
Number of cylinders	6
Capacity (tons)	14
Air-fuel ratio	20:1

Tunnel	A	A	C	B	B	C
Locomotive number	1	2	3	3	4, 5	6
Engine load	Idle	Idle	Idle	Load	Load	Load
Carbon monoxide concentration, per cent:						
Manifold			0.15	0.10	0.10	0.15
Exhaust			0.035	0.03	0.03	0.03
Tunnel air: maximum	0.0055	0.0025	0.0020	0.0025	0.0040[a]	0.009
Minimum	0.0045	0.0000	0.0001	0.0010	0.0006	0.0001
Rate of air flow, cu. ft. per min	3600	12,000	25,000	25,000	45,000	18,500
Air movement, ft. per min	15	50	85	85	200	70
Reactions of men	Poor	Fair	Good	Good	Good	Good

[a] This sample was taken downstream of both engines, with one engine idling only 15 ft. away from sampler, and is not representative.

TABLE 3.—*Results of Tests on Diesel-driven Trucks Inside and Outside of Tunnel G*

Truck No.	Capacity, Tons	Horsepower	Number of Cylinders	Cycles	Where Tested	Grade, Per Cent[b]	Carbon Monoxide in Exhaust, Per Cent
1	15	150	6	4	Outdoors	idling	0.016
1	15	150	6	4	Outdoors	20	0.015
2	8	150	6	4	In tunnel	idling	0.018
3	8	130	4	2	Outdoors	idling	0.031
3	8	130	4	2	Outdoors	20	0.045
4[a]	8	150	6	4	In tunnel	idling	0.02
4[a]	8	150	6	4	Outdoors	20	0.11
5	8	150	6	4	In tunnel	idling	0.015
5	8	150	6	4	In tunnel	10	0.035
5	8	150	6	4	Outdoors (closed cab)	20	0.085

[a] These trucks had been in continuous service for about 12 months.
[b] No discomfort noted, but visible smoke was emitted during pull up grade in tunnel.

trucks were tested in the tunnel, out of doors when idling, and when operating on a 16 per cent grade (Table 4). The trucks were used to remove blasted rock from the face of the tunnel and were required to travel up a ramp to discharge their contents on a cull pile. The engines were six cylinders and delivered 85 hp. When the trucks were operated in the main tunnel (35 ft. in diameter) they were required to travel a distance of 300 ft. The results of tests made on the final exhaust are presented in Table 4. The condition in the main tunnel was good and there was no noticeable smoke or mist. The air movement at the center of the tunnel 6 ft. above the bottom ranged from 80 to 100 ft. per minute. In Table 4, the con-

TABLE 4.—*Results of Tests on Diesel-driven Trucks in Tunnel E*[a]

TWO TRUCKS IN TUNNEL AT ONE TIME; 85 HORSEPOWER, SIX-CYLINDER ENGINES

Engine No.	Load, Tons	Grade, Per Cent	Carbon Monoxide in Exhaust, Per Cent
1	Idling		0.02
	13	16	0.01
2	Idling		0.02
	13	16	0.02
3	Idling		0.01
	13	16	0.03
4	Idling		0.01

[a] These trucks were working in an open-portal tunnel, about 400 ft. from the portal. No visible smoke or discomfort was noted. Air movement, 100 ft. per min. (80,000 cu. ft. per min.).

centrations of carbon monoxide approximates those given for the Diesel locomotive in Table 2. Another series of tests made on five trucks operating in a tunnel on 10 and 20 per cent grades is given in Table 3. These trucks, with one exception, were six-cylinder trucks rated at 150 hp.; the other truck had four cylinders with a 130-hp. rating. A few of the concentrations were high. This is attributable to the fact that the engines causing these concentrations had been in continuous service for about 10 months without extensive overhaul.

Bulldozer Tests.—The bulldozer tests were made on a caterpillar tractor driven by a six-cylinder Diesel engine rated slightly over 100 hp. The tests were conducted in a tunnel supplied with 10,000 cu. ft. of air. The concentrations of carbon monoxide obtained in the breathing zone of the operator and in the final exhaust of the engine are shown in Table 5. The concentrations in the tunnel air were relatively low.

TABLE 5.—*Results of Field Tests Made on Diesel-driven Caterpillar Bulldozers*[a]

Rated horsepower.......... 108.3
Number of cylinders......... 6
Air-fuel ratio............... 22.3:1

Tunnel..........	A	A	D	F	F
Diesel caterpillar No...........	1	2	3	4	4
Engine load......	Idle	Full	Full (16 per cent grade)	Full	Idle
Carbon monoxide concentration per cent:					
Manifold....				0.050	0.034
Exhaust.....	0.012	0.020	0.01	0.020	0.011
Tunnel air...	0.005	0.005	0.000	0.004	0.001
Rate of air flow during tests, cu. ft. per min.....	30,000	40,000	45,000	10,000	10,000
Air movement, ft. per min........	100	150	150	50	50

[a] No smoke or discomfort expressed by workers or noted by investigators.

SUMMARY AND CONCLUSIONS

The Labor Department has conducted extensive hygienic studies on Diesel engines under actual operating conditions in tunnels. Trucks, bulldozers, and locomotives driven by these engines have been tested during all typical tunneling operations. They have operated from both shafts and portals and in dead-end runs and tunnels open at each end.

The conclusions that can be drawn from the test data obtained during these studies can be summarized as follows:

1. Diesel-powered machinery can be safely and satisfactorily used underground if careful adherence to certain strict regulations is maintained.

2. The New York State Labor Department has issued such regulations to govern the use of Diesel engines in underground operations in New York State.

3. During all types of operating conditions, these regulations have been found to stand the test of practical application and adequate control.

4. The first section of requirements covers the design and maintenance of the engines. The main points of this section are: (*a*) 20:1 minimum air-fuel ratio, (*b*) the exhaust gases must be cooled and scrubbed, (*c*) before discharge the exhaust must be diluted at least 10 times.

5. The second section covers allowable limits of contamination. Careful correlation of all factors has shown that the control of carbon monoxide to the set limit of 0.002 per cent affords adequate control of all noxious element.

6. The third section is devoted to requirements of ventilation, which have been found adequate to provide the required control. The governing factor in this section requires that a minimum of 10,000 cu. ft. per min. of mechanical ventilation be maintained in any area for every engine that is or may be operated in that area.

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Multiplying Manpower with Scrapers

By Roger V. Pierce,* Member A.I.M.E.

(New York Meeting, February 1943)

Introduction

In the last few years, much study has been devoted to increasing stoping efficiency. The reasons for this are shortage of manpower, shorter working hours, operating regulations, and shortages of essential materials. More efficiency in mining methods is needed to offset these adverse factors. In the past, attention has been centered upon haulage, hoisting equipment, mill flowsheets, etc., but recently the spotlight has been focused on the cost of stoping and the cost of handling materials. Scrapers and scraper hoists have thus assumed greater importance. In order to make it possible for certain ore bodies to be mined at a profit, it has been necessary to apply assembly-line practices to the streamlining of mining methods. All cycles of the mining operation must balance, and the sooner broken ore is removed from the working face, the sooner another round can be started.

Since the introduction of scraper hoists in metal mines more than thirty years ago, much progress has been made in their design and application. Alert operators are now constantly watching for methods in which to apply scrapers. The saving of manpower in development work and the changing of mining methods that once employed manual handling of materials are increasing vital wartime tonnages.

Essential ores from newly discovered ore bodies are moving rapidly from stopes to reduction plants—because it is now possible to develop an ore body and place it in production with a minimum of manpower and development work. Today, speed is one of the prime essentials of mining.

Everywhere, there is a concentrated effort to use experienced miners in breaking rock, in timbering, and in carrying out the various phases of mining that require many months of training. Much effort is being made to mechanize the working places because experienced men naturally prefer to work where the majority of materials are handled mechanically. In line with this policy, many new men are being taught the details of scraper work, because even green men can quickly become acquainted with scrapers and thereby release older heads to perform the more skilled phases of mining.

In some places it has been found economical to change from one system of stoping to another that more readily allows the application of mechanical muck-handling equipment. Where blocks are worked faster, the cost of *materials*, *time*, and *labor* is cut drastically. This favorable situation allows more manpower to remain on straight breaking and timbering—or straight production mining. Moreover, when blocks are mined faster, more floors can often be mined vertically before the mucking or wheel floor is moved. This results in several direct savings.

There are cases where it has been proved that by working a given block faster more area, longitudinally or vertically, could be mined at one time, thus permitting the reduction of development cost. (Fig. 1.)

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In areas difficult to mine, scraping has made it possible to mine faster and to keep vital filling or gob packed close to the mining floors. Fundamentally, the moving of muck in a stope, intermediate drift, or crosscut involves a loading operation into a car or a wheelbarrow, either by hand or machine, and then a tramming operation to a dumping point. There are, therefore, two separate operations; namely, *Loading and Tramming*. With scraping, where distances permit, these two operations are combined. The properly designed scraper travels empty to the ore pile and loads without help from the operator or from anyone at the loading point. The loaded scraper is then pulled to the dumping point where its load is discharged automatically.

FIG. 1.—SQUARE-SET STOPE IN EXTREMELY HEAVY GROUND.
This area was safely and quickly mined with a small scraper hoist.

Slushing has developed the present self-filling, self-dumping, bottomless-type scraper that is operated by air-driven or electric-driven two-drum or three-drum hoists. There are units on the market built to fit every known underground mining condition. There are hoists that are definitely portable; in fact, hoists weighing little more than a standard drifter drill. These small hoists handle 12 to 17 tons an hour from a distance of 50 ft., and their pull distances range up to 100 ft. There are hoists of corresponding rope speeds and pulls that make it economical to transfer broken ore through distances of hundreds of feet.

The ideas in the following pages will supplement those in general use and will point out more recent applications, operating ideas and shortcuts, as well as developments in hoists and mountings. This paper is presented from the standpoint of practical mine operating. The diagrammatic sketches and pictures illustrate these points.

There are many mines in the country that scrape all of their underground broken rock. Some mines have changed their stoping system from one method to another in order to permit full savings in production with scrapers, and there are some whose system allows the use of only a hoist or two. It is difficult, however, to point now to any mining system that does not have a certain amount of work that can be handled by scrapers. A study of stoping methods will show that there are scraper applications in every one.

Development Work

Scraper hoists have an application in virtually every type of mine heading. The more profitable applications of scraping in development work are those where intermediate drifts or crosscuts are being driven from a raise. There are instances where a lower level has been developed beyond work completed on the level above. If haulage and switching facilities are limited, it is very economical to use a scraper hoist to pull the level develop-

ment rock to a raise. There are several places where this has been done. (Fig. 2.)

This type of application applies to development work, especially where waste

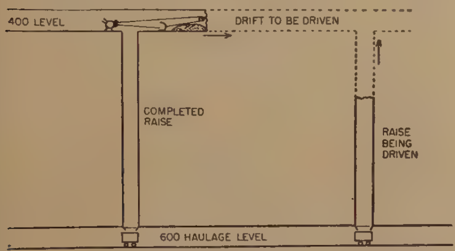

FIG. 2.—DEVELOPING LEVELS WITH SCRAPER HOISTS WHERE RAISES ARE AVAILABLE FROM LOWER LEVELS.

for stope filling is developed by driving waste crosscuts out into the hanging wall or footwall of the vein. This has become more feasible since the introduction of the small, 250-lb. air-driven slusher hoist.

Intermediate drifting or sublevel work is an ideal application for such types of equipment, both from the standpoint of speed and the standpoint of cost per ton or cost per foot of development.

Short, small crosscuts can be driven quite cheaply with light, portable equipment. Where large development headings are driven, the larger "cat-mounted" ramps are now giving low-cost tonnage figures. These devices are powered with air or electric motors and are equipped with the larger three-drum hoists that handle from 50-in. to 70-in. scrapers. They are suitable in headings 10 ft. or more in width, for in these places more tonnage is available. Since the headings are wide, the three-drum hoist eliminates the necessity of double tracking, which is required by shovels working in low headroom.

Station Cutting

Scraper hoists have been used in cutting stations at a number of places. Two of the more popular methods are shown in Figs. 3 and 4 and are described below.

Fig. 3 shows the shaft completed to below the level of the station, the skip pocket raised from the shaft to the sill of the proposed station, and a small pilot crosscut driven from the shaft to the

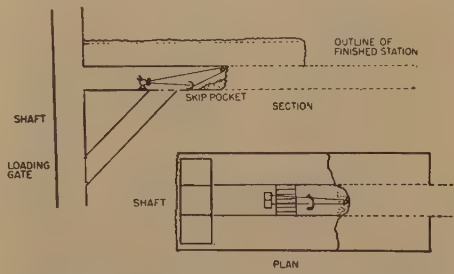

FIG. 3.—ENLARGING AND CUTTING STATION TO SKIP POCKET WITH SCRAPER HOIST AND SMALL PILOT HEADING.

top of the skip pocket. All muck from the station-cutting job and early-level development is scraped directly into the skip pocket.

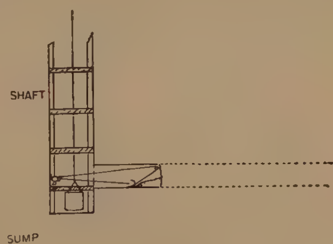

FIG. 4.—CUTTING STATION AND STARTING EARLY DEVELOPMENT WORK ON LEVEL WITH SCRAPER AND SMALL PORTABLE SCRAPER HOIST PLACED ON WALL PLATE IN SHAFT.

Fig. 4 shows a station-cutting project employed where a skip pocket is not used. In this case, a small, two-drum air hoist is placed on the wall plate in the shaft, and the broken muck is pulled directly into the shaft-sinking bucket. The bail of the sinking bucket can be seen below the hoist in Fig. 5. This picture shows a small, lightweight scraper hoist mounted on a wall plate in a vertical shaft. This is used in conjunction with a 28-in. scraper to cut an underground station. The broken rock is pulled to a short slide chute, where

it is delivered into a sinking bucket in the shaft.

Flat raises are also an application where costs have been greatly reduced by this

Fig. 5.—Lightweight scraper hoist in conjunction with 28-inch scraper to cut station.

type of equipment. There are any number of places where underground pump rooms, sumps, and other openings have been completed by the use of scrapers.

In another interesting case, a vertical shaft was planned to connect a main-adit haulage level and several of the lower operating levels. The shaft site was spotted and a single compartment raise driven from the lower level to the adit level. The raise section was enlarged with the aid of long drill holes that paralleled the raise. All broken muck from these various sections was pulled off by a scraper on the operating levels, the muck being taken through a short crosscut. The scraper dumped directly into cars, as shown in Fig. 6.

Filling

In many mining methods, filling is as important as the actual ore extraction. It is extremely important to have mobile equipment for this type of work. It has been proved that the use of a small scraper-hoist unit, especially if the distances are

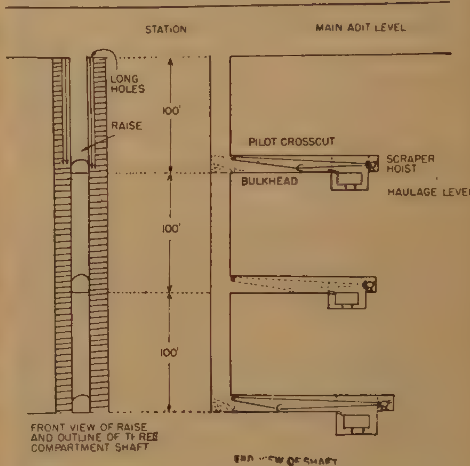

Fig. 6.—Enlarging raise with long drill holes to form three-compartment vertical shaft.

short, will result in marked economies. In cut-and-fill, square-set stoping and other methods, the hoist handling the ore to the chutes is also utilized in handling the back filling. Every element of speed and all possible shortcuts are used in removing the ore, filling the area, and keeping a steady flow of ore headed toward the reduction plants. All this is accomplished without any more repairing or timber replacing than is absolutely necessary.

Transfer

Transfer and short haulage constitute another vital function of scrapers in the present-day systems of mining. In many installations scraper transfers are eliminating cars, tracks, hand tramming, and chute loading. Tandem scraping has increased the economical limits of ore haulage by this method. This has served to eliminate costly chute installations and is another means of saving manpower and vital materials. Transferring has also

eliminated hundreds of feet of costly raise development.

There are places where sticky, claylike material or large, heavy boulders are encountered. Under such circumstances it has paid to eliminate chutes and chute men from the main haulage levels. When material is sticky or breaks large, it is costly to tie up a motor, a train of cars, chute pullers, and motor crew while a chute is being freed. To prevent this, the chutes are brought down vertically with a short offset into a crosscut, which has been driven at right angles to, and slightly above, the haulage level. An electric hoist is used to pull the loaded scraper from the chute terminal directly over the car. When the train comes in to be loaded, very little time is expended in the usual chute-mouth loading delays. This eliminates uncontrolled chute runs and surges that often are encountered in materials not handled easily by conventional chute-loading methods. One such method is shown in Fig. 7.

Safety First

There has never been a period when a lost-time accident could cost so much, and be so harmful to production, as today. Through the introduction of transfer scrapers, a large percentage of raise networks has been eliminated. This automatically cuts down on a type of work that usually has a high accident rate.

In heavy ground, the worker remains under untimbered backs only a fraction of the time he would be there if he were using hand mucking methods or mechanical methods requiring the operator to be at the face. There is little chance for accidents caused by rocks rolling down from the pile onto a mucker's foot. The mucker is a safe distance from many of the sources of possible danger—"he's in the clear." When sublevel headings are driven in ground requiring timber, the round of muck can be cleaned out or pulled back with scrapers, and the timber can be stood before the ground takes weight and starts sloughing.

In stoping requiring fill, especially those

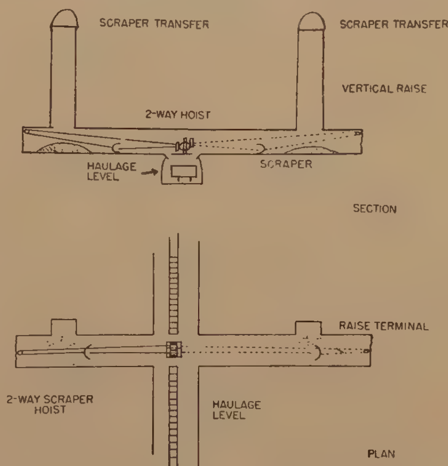

Fig. 7.—Scraper hoist applied to transfer operations.

on the borderline between timbered and untimbered stopes, it is not necessary for the miner to work under ground that has remained open after the cut has been made. This also holds true after the ore has been cleaned out and a high, unsupported back is left. This work can be done rapidly without risk to anyone. Stoping hazards are reduced to a minimum.

Scrapers Applied to Existing Mining Systems

Mining Irregular Ore Bodies

There are cases in which the introduction of scrapers would not lead to economical operation if older types of equipment were used; for instance, when tonnages per working place are small. For scrapers to be feasible, something light and portable had to be available. A scraper hoist having these features was placed on the market about five years ago.

Consider, for example, a situation where the ore body was broken by a series of faults. The individual ore blocks were

small, and obviously the tonnage per block would not justify the time and labor required to move heavy equipment into such areas. The savings effected by the

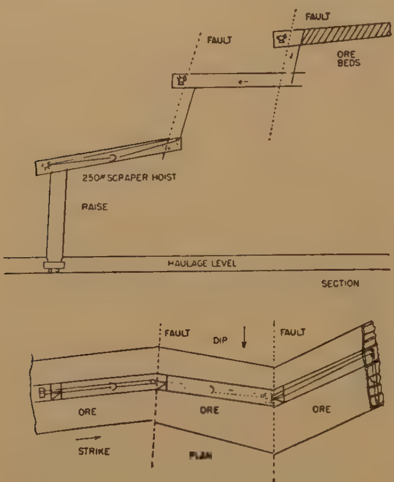

FIG. 8.—RECOVERY OF ALL ORE IN BLOCK BY SCRAPERS.

use of large scrapers were lost in moving the equipment to another ore block. The lightweight, air-driven double-drum hoist handling 1¼ wheelbarrows per trip at 125 to 150 ft. per minute worked out very favorably in a number of these stopes.

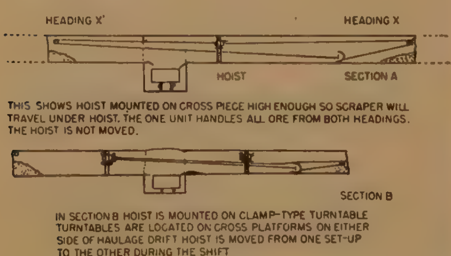

FIG. 9.

Figs. 8, 9 and 10 illustrate some of the problems encountered, and the methods introduced to overcome them. Fig. 8 shows one of the variations in mining one of the blocks. The small hoists are used not only as production units in the various individual blocks, but also for transfer work.

In Figs. 9 and 10 a section of an ore body is shown where it occurs on about the same plane as the haulage level. Here again, the small hoist is used to mine the

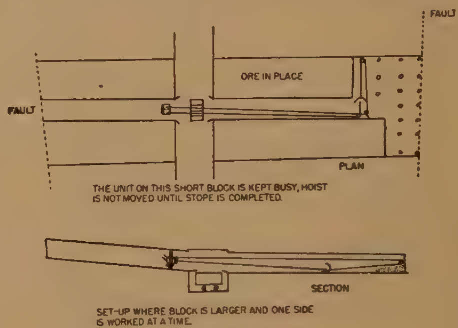

FIG. 10.

stull stope. These sketches show the possible methods by which the small hoist can be mounted.

FIG. 11.—LIGHTWEIGHT HOIST MOUNTED ON SKIDS.

Section A of Fig. 9 shows the hoist mounted on a crosspiece high enough for the scraper to travel under the hoist. This is mounted in such a way that the scraper handles all ore from headings X and X'. The hoist is stationary.

Section B of Fig. 9 depicts the hoist where maximum portability is required. A semipermanent mounting was installed on each side of the haulage level. A clamp-type turntable was installed on each crosspiece, and these were placed high enough so that the scraper could work under the opposite mounting without fouling this mounting. To move from one setup to another required only the removal of the cables from the scraper, the loosening of the swing clamp on the turntable, and the actual moving of the hoist to the location directly across the haulage level.

Fig. 10 shows a block large enough to warrant keeping the hoist in one place for some time. This is: (1) a standard setup; or (2) one where the hoist is mounted on a timber base that is held in place either with stulls or with a vertical column bar; or (3) a setup that is somewhat similar to that shown in Fig. 11, where the small hoist is mounted on skids and is moved from one place to another under its power. When moved to a new working location, the layout is held rigid with the aid of a vertical screw-type drill column as shown.

Timbered Cut-and-fill Stoping

At one place scrapers were given a careful study on one or two types of work, and eventually they were used in all phases of stoping operations. Once proved, the entire stoping system was changed to take advantage of the savings scraper equipment offered. (Figs. 12, 13 and 14.)

The system follows timbered cut-and-fill, or a variation of stull timbered stoping. Raises are placed on 100-ft. centers, and the block is mined by cutting successive floors through from one raise to another. Until the advent of scrapers, ore was broken down onto the mucking floor, and then hand-mucked through drop boards directly into cars standing on tracks on the wheel floor below. As these floors were worked out, waste was obtained from crosscuts driven into the footwall or hanging wall. This was brought into the stope with wheelbarrows. With the new method all ore is now handled by 7½-hp. air slushers and 30-in. steel scrapers and the waste is moved into the stopes, where it is spread to the various sections with a small, portable hoist.

This is an example where all broken rock in the stopes is now handled by scrapers. Cars and tracks have been removed from the stopes. Hand tramming has been eliminated. Considerable material is saved, because it is not necessary to build track requiring ties, cross-sill timber, lagging, etc.—material that is not recovered when the stope floors are raised.

In Fig. 12 the gob section is kept fairly close to the mucking floor. Very little open ground is left below the mucking floor. With both hoists working in conjunction, the stoping cycle is quite smooth. The mucking crew keeps the areas clean, so that there are always places where the drillers can keep working, and there is a steady flow of ore to the chutes during each shift.

One 7½-hp. hoist handles ore from two faces in block A and one face in block B. The hoist is mounted on a turntable, so that it can be lined up with either line of timber in the stope.

To muck out rounds of ore broken behind the hoist or in block B, the scraper is pulled underneath the hoist to the chute dumping position. See Section A of Fig. 9 for a similar setup.

Untimbered Cut-and-fill Stoping

In untimbered ground, the small, double-drum air hoist is used in horizontal cut-and-fill stoping. Raises are about 100 ft. apart. The horizontal cut is taken from one raise to the next. The small hoist is mounted on a horizontal drill column, and the scraper passes under the bar, thus permitting the operator to stand in the center

of the stope and see the scraper at any time during its period of travel.

Units of this type, using 235-lb., 28-in. scrapers, handle about 500 to 600 lb. of ore per trip. This equals 12 to 17 tons per hour up to 50 ft. and the hoists are used on many types of work up to distances approximating 100 ft. The rope speeds run between 125 and 150 ft. per minute with air pressures of 80 to 90 lb. The setup shown in Fig. 15 and in Section A, Fig. 16, allows the operator to scrape ore into the chute at either end of the block. This assures a very economical operation, since the hoists use only about as much air as a 45-lb. Jackhamer.

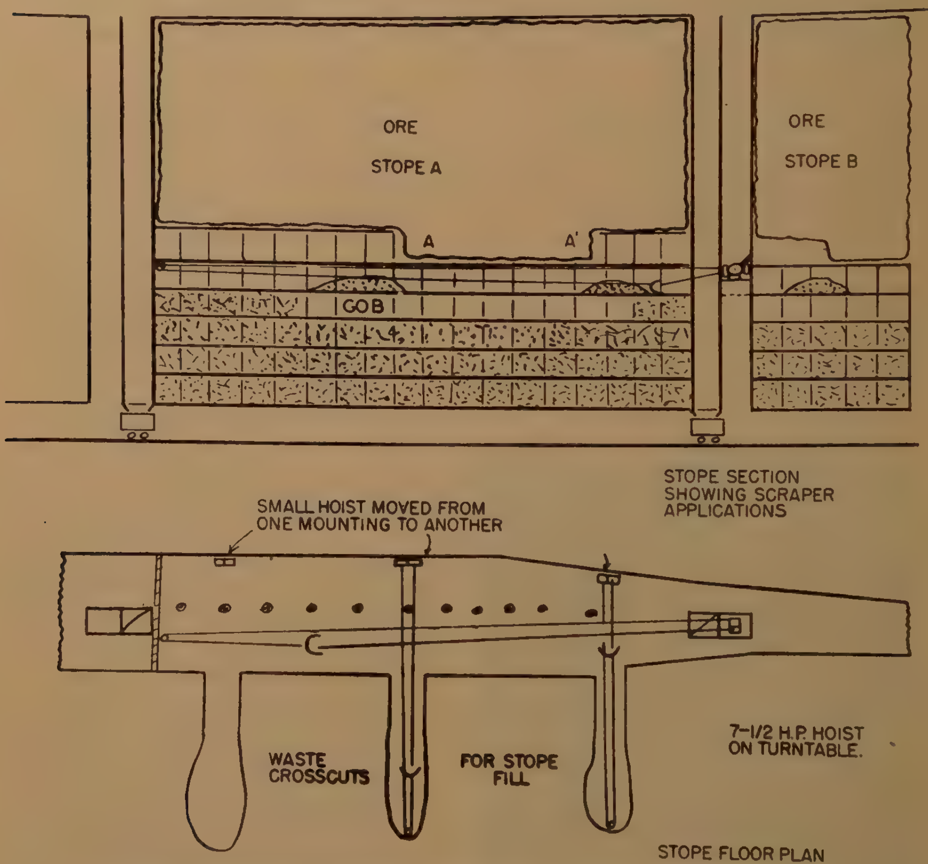

Fig. 12.—Example of timbered cut-and-fill stoping with scraper hoist.

After a slice has been taken from one raise to the other, the ore is slushed to the raises and the area is filled with sand. When filled to the proper level for the next cut to be taken, the mining end of the cycle is started once more.

Combinations and Variations of Underhand Stoping Cut-and-fill

Various sizes of hoists are used, the specific unit depending upon the tonnage and the length of the job involved. Described are two cases in which the small hoist is utilized in carrying out all materials-handling operations in a horizontal cut-and-fill stope.

FIG. 13.—GENERAL VIEW OF FLOOR IN TIMBERED CUT-AND-FILL STOPE.
Hoist is cross-scraping waste from a waste crosscut into fill area of stope. Picture shows only one half total width of stope.

FIG. 14.—LOOKING DIRECTLY INTO WASTE CROSSCUT ON MUCKING FLOOR OF STOPE AS SHOWN IN FIG. 13.

The operation is standard cut-and-fill in a narrow vein. The mining cut is made from one raise to the other before the ore is pulled to the chute, and is then made ready for the introduction of the waste fill. This makes possible certain flexibilities, for there are places where cut-and-fill could be worked in conjunction with a modified shrinkage system. For example (as in Section B, Fig. 16), take two, three, or four cuts vertically, or whatever the ground will stand; scrape out the overbreak after each blast; then, when the vertical stoping limit has been reached, scrape out the ore and introduce filling with the speed that can be attained with scrapers. There is very little danger involved, since the workers are not required to spend any time in the open stope when scraping out ore or when filling the mined section with waste. This saves the time ordinarily spent in laying floors between the waste fill and each cut made across the back of the stope. Where such a method is possible, it results in a mining combination that is something between horizontal cut-and-fill stoping and shrinkage—a combination that permits the operator to benefit from the advantages offered by both mining systems.

Fig. 15.—Small hoist mounted on a short horizontal bar in a cut-and-fill stope. Work done is shown in section A, Fig. 16.

Underhand Stoping

Often there are places where a small body of ore can be recovered by underhand stoping methods aided by a scraper and a two-drum or three-drum hoist. One such installation is shown in Fig. 17, which illustrates another variation of underhand stoping. The introduction of scrapers

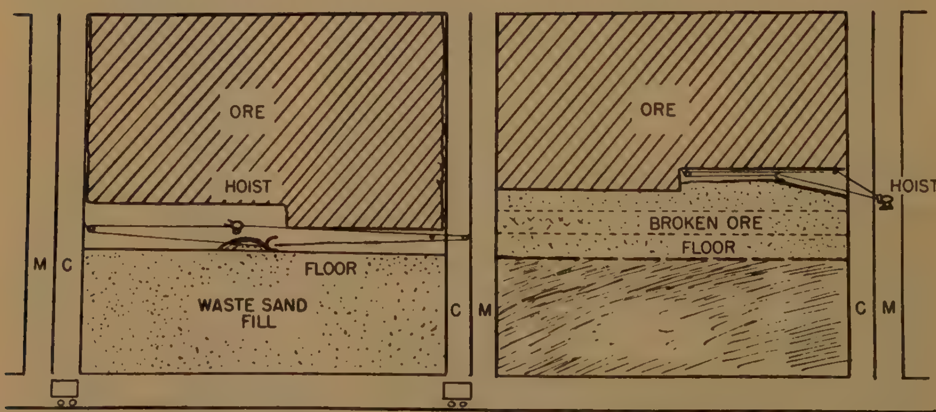

Fig. 16.—*A*, horizontal cut-and-fill using sand filling; *B*, combination cut-and-fill and shrinkage.

makes possible the pulling of all ore to one central chute, thereby making a place for drillers to work each shift and establishing

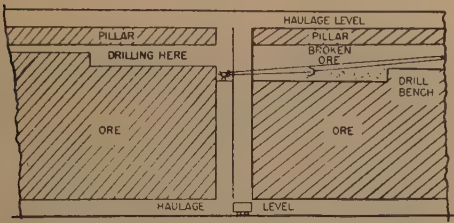

FIG. 17.—ONE VARIATION OF UNDERHAND STOPING.

a balanced working schedule for the miners and mucking crew.

Block Caving

Scraper hoists are being used to great advantage in various ways in the block-caving system of mining. In one application the small hoist is used in cutting through from a branch of the control raise to another in driving the grizzly drift. In this case the raises are 20 to 25 ft. apart. Formerly, 10 to 15 ft. of the distances between these control raises had to be hand-mucked. The small hoist is moved around from one grizzly drift to another and is used to pull back the broken rock a short distance into the last raise. This is shown in Fig. 18.

Another popular idea in block caving is the introduction of the hoist to transfer work. This application substantially reduces the amount of raise development work usually expended in developing a block of ore for extraction. The hoist can be used in the grizzly drift to pull ore from the fingers directly into chutes or into cars on the haulage level. Fig. 19 shows a deposit that dipped below the main haulage level. All ore from this area was loaded directly into cars with a scraper hoist.

The flexibility of the scraper hoist outfit is demonstrated in many places in gathering work, and particularly in transfer work in large areas. Large tonnages are handled by transfer hoists in top-slicing, sublevel caving, as well as in block caving. In each case, numerous elaborate chutes and chute mouths are eliminated,

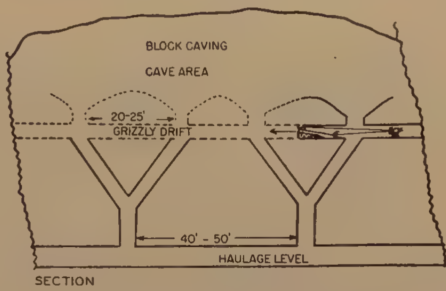

FIG. 18.—SMALL SCRAPER DRIVING GRIZZLY DRIFT IN BLOCK CAVING.

thus concentrating ore in a few places and eliminating the problems of multiple-chute loading, which accounts for serious train delays. Cars, rails, and hand trammers are removed from the sublevel

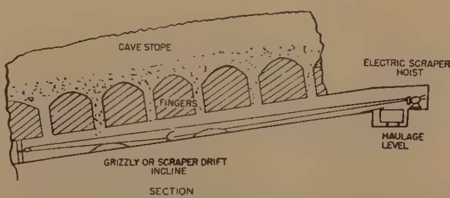

FIG. 19.—USE OF SCRAPER FOR TRANSFER WORK IN BLOCK CAVING.

transfer areas. Costly development work is cut to a minimum.

Incline and Flat, Room-and-pillar Stoping, Breast Stoping

There are many applications for scrapers in flat and incline bedded deposits. The absence of gravity flow necessitates moving ore to the haulage levels by mechanical means. There are flat, bedded deposits where rubber-tired haulage in connection with "cat-mounted" scraper loaders are being worked very economically. The "cat" mountings are very flexible and are one means that can be used in moving an outfit over irregular floor surfaces. The scraper, too, will follow the bedding planes of an ore body. These flexible units will

fit unforeseen mining conditions. Rubber-tired outfits fabricated from heavy truck bodies are also being employed successfully.

Fig. 20 shows a 25-hp. three-drum hoist mounted on a "cat-driven" loading ramp working in a horizontal room-and-pillar stope. This is used in conjunction with rubber-tired haulage units. The 48-in. scraper used in this case is an all-cast-steel, semibox, hoe type. Four or five trips are required to load 2 tons of ore.

There are outfits placed on stationary mountings (such as shown in Fig. 21) that are used to pull broken ore down the slopes directly into cars on the haulage level. Fig. 22 shows the loaded scraper coming

Fig. 20.—Three-drum electric hoist mounted on "cat-driven" loading ramp.

Fig. 21.—Two-drum hoist on haulage level directly in front of a loading chute. This is an incline room-and-pillar operation.

down the inclined stope floor. The same applies to underhand stoping in dipping ore bodies.

In the operation shown in Fig. 23 the vein dips 20°, and for the most part is about 3½ ft. thick. In some sections the thickness reaches 5 ft. The foot and hanging walls are solid enough to permit incline room-and-pillar stoping methods. Fig. 21 shows one way of mounting the hoist. The stope is large enough so that the hoist can be mounted on the sill, where it remains until the area is mined out. In other areas, a hoist is mounted on a built-up car and this is moved from one chute mouth to another. See Fig. 24. This is a very flexible arrangement. During the time the drillers are working in one stope, the hoist is at another stope entrance loading broken ore, thus permitting the hoist to be in almost continuous operation.

FIG. 22.—4½-FT. SCRAPER PULLING BROKEN ORE DOWN A SLOPE TO THE HAULAGE LEVEL.

Sublevel Stoping

There are a number of variations in sublevel stoping. In certain conditions sublevels are placed fairly close together on the vertical to take advantage of short blocks of ground and the short drill holes needed to break these blocks. There are places where the sublevels are driven midway between the footwalls and hanging walls of the vein, to make available ring-drill stoping methods. Sometimes a sublevel is driven along the footwall and a cut taken toward the hanging wall, so as to form a bench from which vertical holes can be drilled. Sometimes two sublevels are driven on the same elevation and follow approximately along the footwall and hanging-wall sides of the vein. This system gives the drillers a semi-ring drilling setup from which to break the ore pillars. Sometimes blast-hole stoping is practiced, the main sublevels being driven with large scraper hoists. The small coyote holes

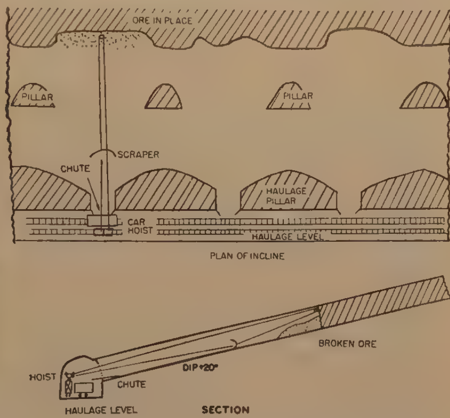

FIG. 23.—ROOM-AND-PILLAR MINING USING SCRAPER HOIST.

FIG. 24.—ONE TYPE OF MOUNTING USED TO MOVE SCRAPER HOIST FROM ONE CHUTE OR SLIDE MOUTH TO ANOTHER.

and smaller entries that are prepared for powder blast are driven with the smaller type of scraper hoists. Whenever these subs can be driven fairly straight, the application of scraper hoists assures speedier production at less cost. (See Fig. 25, Sections A, B, C and D.)

With proper scraping equipment—that is, larger, faster hoists pulling larger scrapers of greater capacity—it is possible to drive longer and larger sublevels at a very reasonable cost. Under favorable conditions, by increasing the cross section of the sublevel and by pulling larger rounds, the cost per ton of ore drops to the point where sublevels can be placed closer together. Such a plan reduces the length of the drill holes needed for breaking the blocks of ore between these subs, thus permitting a saving in drilling costs.

The scraper system permits the sublevels to be driven faster, cheaper, and with less manpower. Great savings are realized by using large transfer scrapers to pull all ore from the bases of the various fingers to one central loading chute. This is true where blocks have been undercut, belled out, grizzly drifts driven, grizzlies placed every 20 to 25 ft., and the raises installed between the haulage level and these grizzlies. In addition to the savings made by reducing undercut and development work, and by pulling all ore to one raise, greater speed on the haulage level can be obtained. Motor crews come to one chute to load all the ore from a block. They do not have to waste time going from one chute to another. Furthermore, it is economical to build one specially designed, fast-loading chute to achieve maximum

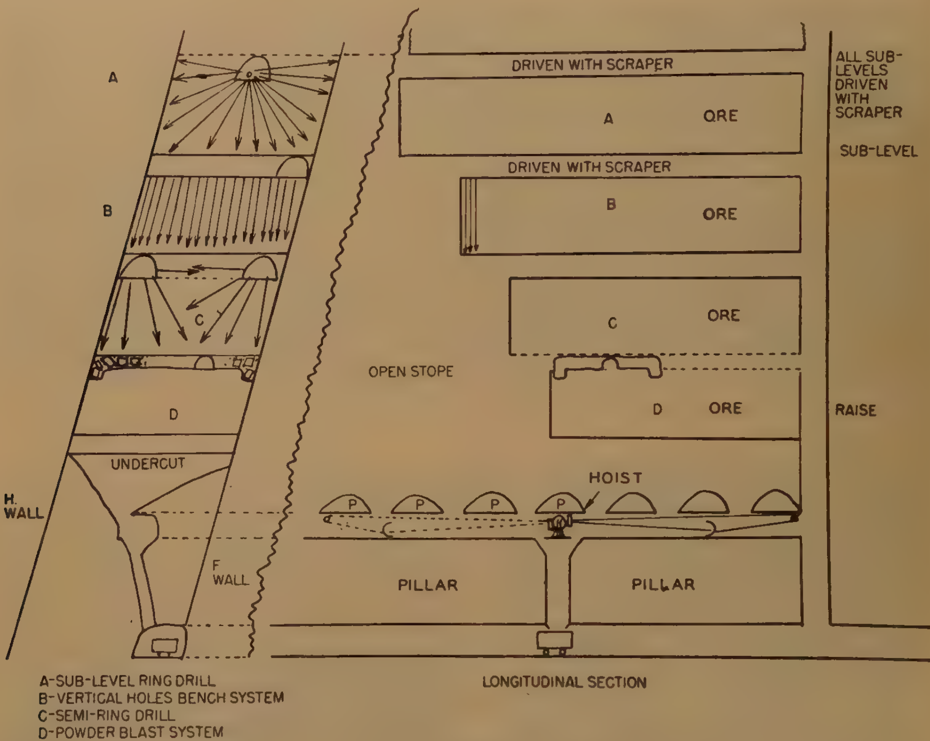

FIG. 25.—COMPOSITE EXAMPLES OF SEVERAL SUBLEVEL STOPING SYSTEMS USING SCRAPER HOISTS.

loading efficiency. With numerous chutes, it is not feasible to spend the money on elaborate chute construction.

Top Slicing or Sublevel Caving

Scraper hoist equipment is today handling practically all of the ore obtained in mines using top slicing or sublevel caving methods. About the only change or addition to these systems made in the last three or four years has been the introduction of the small scraper hoist into areas having limited tonnages. These areas include short blocks, narrow veins, and poorly accessible sections. Where small blocks were to be mined or small tonnages handled per working shift, this small unit has been well received. Figs. 26 and 27 show the small hoist in a western mine in a top-slice stope. The small hoist in Fig. 26 is operated in a top-slice stope pulling mercury ore (cinnabar) to a central chute. The hoist is pulling a 28-in. scraper; the maximum pull distance is 100 ft. From these distances, the outfit handles 6 to 10 tons per hour. Note the roller used to protect the pull cable.

FIG. 26.—USE OF SMALL SCRAPER HOIST IN A TOP-SLICE STOPE.

Shrinkage

There are some special applications of scraping in standard shrinkage stopes. Where the ore body rakes out beyond the raise-developed areas, the ore from this section can be moved economically, and at a cheaper over-all cost than would be required if raises were driven from the haulage level up to the base of the ore body on the usual 20 to 25-ft. centers. When shrinkage stopes are pulled, some ore has a tendency to funnel, making necessary the use of a hoist in leveling the muck pile so that subsequent cuts can be drilled from the back. Hoists are also used in removing all broken ore that hangs up in the stope after the block has been mined. Sometimes a small unit can be effectively used in following small stringers out into either wall, or for general exploratory work. Hoists are used for transfer work under shrinkage blocks.

Mitchell Slicing

Operators employing Mitchell slicing are finding the small hoist profitable and handy in moving ore from the slots to the main

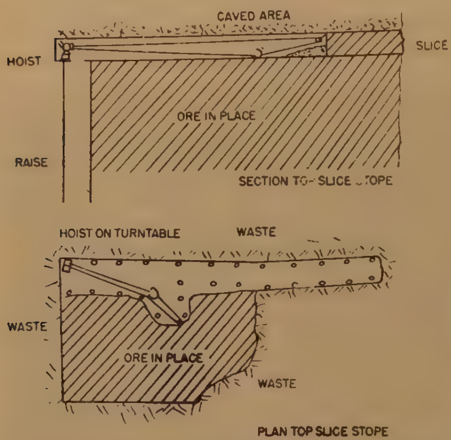

FIG. 27.—TOP SLICING WITH SMALL SCRAPER HOIST.

transfer and also in back-filling mined-out areas.

Scraper hoists of various sizes are popular for transfer work in the Mitchell slice system. Their application in this system, as in many other methods, allows greater areas to be mined from one central raise.

Square-set and Stull-timbered Stoping

The small hoist has had a more popular reception in square-set and stull-timbered stoping systems than in any of the others mentioned. The portability of the small hoist is its most important feature because it quickly and easily solves the problem of gathering ore from a number of places during the working shift. The hoists are used both for handling all broken ore to a central chute and for moving fill into the worked-out sections of the stope. In this system of mining alone, dozens of applications and special mountings have been made with the two smaller types of hoists. Figs. 28 and 29 show what can be done where mining is difficult.

In Fig. 28 a narrow vein is picked up in a worked-out and inaccessible section of a mine. The ground is quite heavy, creating a difficult supply problem and relatively dangerous working conditions.

FIG. 28.—SCRAPER HOIST OPERATING IN NARROW VEIN.

The picture illustrates how small scraper-hoist units can be used in heavy, restricted areas. It also shows how it is possible to resort to cross stulls placed as braces at about the centers of the posts to hold the mucking floor open even in heavy ground. The scraper can pass under these low-headroom areas and still effectively complete the job.

Fig. 29 shows a square-set stope in which the scraper is moving waste fill into a gob area. In heavy square-set stoping ground, it is imperative to keep filling as close as possible to the mucking floor. In this case, a very short period elapsed between the time the floor was worked out and this same area was tightly filled with waste.

SUMMARY

The present war effort demands metals far beyond previous production capacities. This unprecedented demand, coupled with

manpower shortages, would create a most serious problem were it not for machinery.

Fortunately the painstaking development of slusher-hoist equipment is substantially completed. The introduction of slushing into mining, with the necessary system modifications, has developed and matured over the years. Consequently, tried and proved equipment is available to mine operators so that they can complete the mechanization of their operations.

FIG. 29.—MOVING WASTE FILL IN HEAVY SQUARE-SET GROUND.

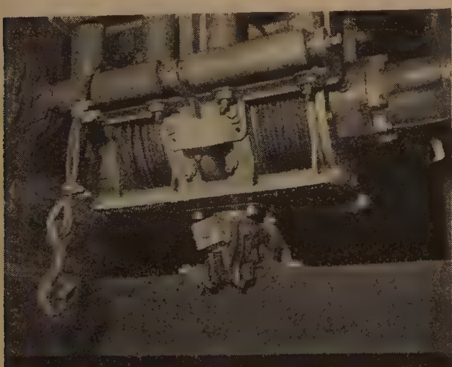

FIG. 30.—PIGTAIL AND CONE-CLAMP MOUNTING FOR HOIST.

The present swing to the use of slushers will doubtless develop to an even higher degree of efficiency, and thus hasten the completion of slusher mechanization underground. Some of the more important advantages are:

FIG. 31.—FLOODLIGHT ANCHOR.

FIG. 32.—TELESCOPIC BOOM.

1. Greater safety.
2. Greater utilization of manpower.
3. Less development work.

4. More tonnage mined.
5. Critical supplies conserved.
6. Conservation of timber, rails, and other equipment.
7. Greater percentage of ore recovered.

FIG. 33.—QUICK-ATTACHING HOOK.

Obviously, the solving of the manpower shortage is the most important benefit offered by scraper hoists today.

Practical Innovations

It is quite important that portable equipment have some quick means of attaching and detaching the pull and pull-back cables from the scraper when the outfit is being moved from one place to another. The pigtail shown in Fig 30 is one type of such connection. To attach or unhook the cable to or from the scraper, it is necessary only to weave the "pigtail" onto the scraper eye.

Fig. 30 also shows a lightweight and inexpensive mounting that was made for a small, double-drum slusher hoist. The mounting consists of the top half of a drifter cone clamp which is welded to a plate that can be bolted to a timber cross-member in a stope. With these turntables located strategically in the stope, it is a quick and simple matter to unloosen the swing bolt, lift the hoist, and move it from one setup to another.

It is advantageous to have good lighting in a scraper stope or transfer. Fig. 31 shows one means of arranging the light so that it can be easily moved from one place to another. It is very similar to an ordinary ice tong. The tong is used to fasten the light quickly to any timber in the stope.

It is often difficult to hold a wedge-eye or breast pin in soft ground. Where timber is not close enough to the breast to anchor any of the hook or chain-type hold-backs, it is necessary to devise a telescopic or adjustable boom. Fig. 32 shows the boom placed between the lead set of timber and the unbroken ore face.

Where scraping is carried out under timber, an anchor or quick-attaching hook facilitates the moving of setups. The "double dogs" with connecting chain shown in Fig. 33 are very practical and successful.

Ventilation of the Climax Mine

By Leo H. Glanville*

(New York Meeting, February 1943)

Until 1934, natural ventilation was depended upon in the mine of the Climax Molybdenum Co. at Climax, Colorado. In that year a 7-ft. axial-flow, low-pressure fan was installed as an exhausting unit. In 1936, three more fans of the same kind were installed, all as exhausting units, with a total capacity of 115 000 cu. ft. per min., and since then fans and sprays have been added as required.

Atmospheric Conditions

Adverse atmospheric conditions at this elevation of 11,500 ft. account in part for difficulty in maintaining adequate ventilation throughout the mine. Barometric pressures range from 18.99 to 20.01 in. of mercury; the normal average being 19.70 in. Air weight is 0.051 lb. per cu. ft., and humidity is low—about a 30 per cent yearly average. This makes the use of mechanical humidity aids mandatory the year around to prevent excessive dryness of the mine workings and the resultant dust hazard.

The yearly temperature range is from minus 24° to 75°F.; typically, in February 1942 the mean low was minus 3.25°F. and the mean high, 21.3°F. In August the mean low was 34.8°F.; the mean high, 64.26°F.

Even during the summer month air filtration through the openings of the cave area is 32°F., owing to accumulation of snow in the glory hole and in part to glacial ice, which, combined with a rock temperature of 39°F. on the upper levels (elevation 11,945 ft.) and 42°F. on the main haulage level (elevation 11,470 ft.), makes it difficult to maintain a comfortable working temperature.

The one advantage in ventilation of mine workings at the altitude of the Climax mine is that the power required to circulate a given air volume is 32 per cent less than would be required at sea level.

Problems

Since 1936 the mine management has studied intensively plans for providing adequate ventilation for the entire mine for its expected life. As the ore breaks into large pieces, much secondary blasting is necessary, therefore there is a considerable amount of smoke, gas and dust, which requires circulation of a large amount of air to remove the contamination.

It is undesirable to use haulage drifts and manways as air passages because some would be passageways for contaminated air also, and there would be undesirable high velocities in some drifts.

High velocities and large volumes of subfreezing air dry out the passageways through which they pass, and thus increase the amount of dust. Regulating doors in haulage drifts are difficult of upkeep and interfere with production.

It was decided as a general principle to distribute fresh air to each working place at low velocities and low pressures and to remove contaminated air as soon as possible with a minimum of travel through working places.

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Fig. 1.—Footwall ventilation.

Fresh air entering this level is through the Buffer tunnel and White level tunnel. From the Buffer tunnel the air split is at 180 ventilation raise, part going down to 705 ventilation lateral (elevation 11,705 ft.) and part up to 940 lateral (elevation 11,940 ft.).

Fans 4 and 5 are exhausting contaminated air from the lower levels. The entire exhaust of 367,000 cu. ft. per min. to the surface is through the main ventilation raise. Fan No. 2 delivers fresh air from the White tunnel to the Phillipson level, about 600 ft. slope distance.

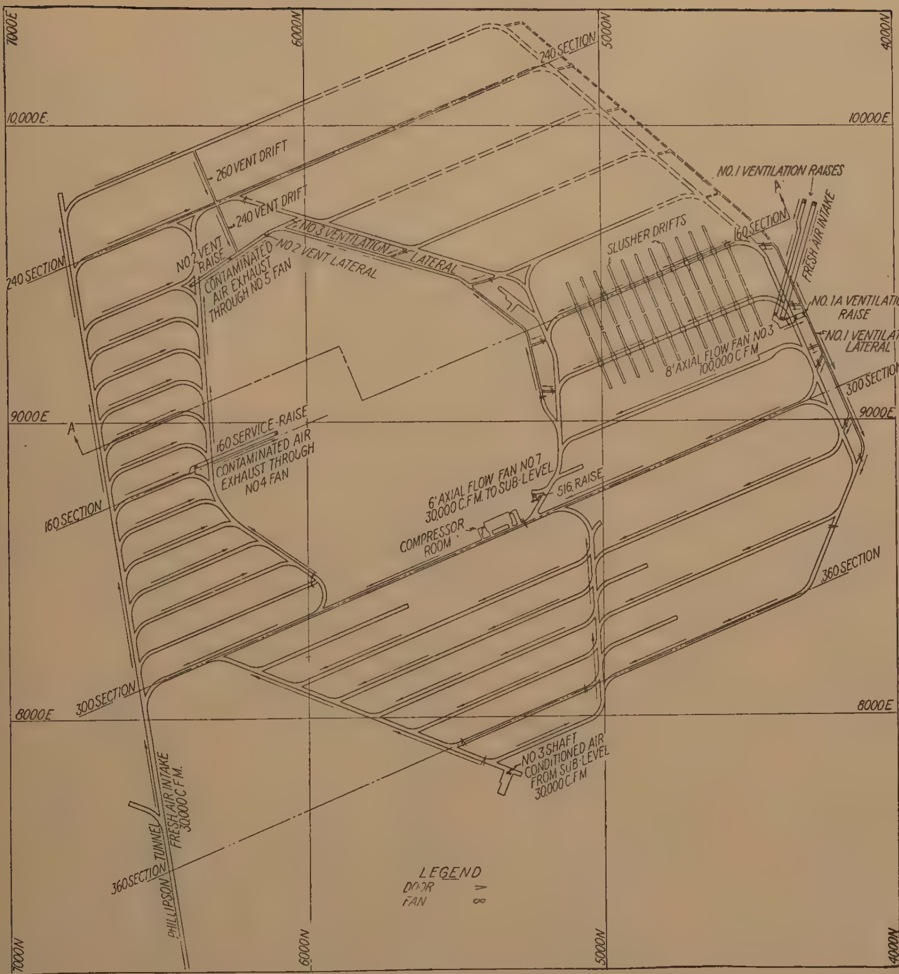

FIG. 2.—MAIN HAULAGE LEVEL.

Broken lines represent undeveloped drifting. Fresh air for this level is from 1*A* ventilation raise, filtration through the cave area and through the main tunnel (Phillipson).

In No. 1 ventilation raise delivery of 100,000 cu. ft. per min. to No. 1 ventilation lateral is effected by an 8-ft. axial-flow fan. Positive control of distribution is by the regulating doors in each lateral entry, allowing any predetermined volume to pass.

Opposite No. 1 lateral at a distance of 920 ft. is No. 3 lateral acting as a return for contaminated air.

The hot-water sprays are at the bottom of 1*A* ventilation raise. All the air entering the mine in that section passes through the sprays, which accomplish the dual purpose of heating and humidifying.


8500 N
9000 E
160 SECTION
240 SECTION
220 N VENTILATION RAISE 12'X12'
CONTAMINATED AIR EXHAUST THROUGH NO 5 FAN
160 SERVICE RAISE 8'x12'
CONTAMINATED AIR EXHAUST THROUGH NO 4 FAN
FOOT WALL
8600 N
260 SERVICE RAISE
5'x7'
FRESH AIR
8000 E

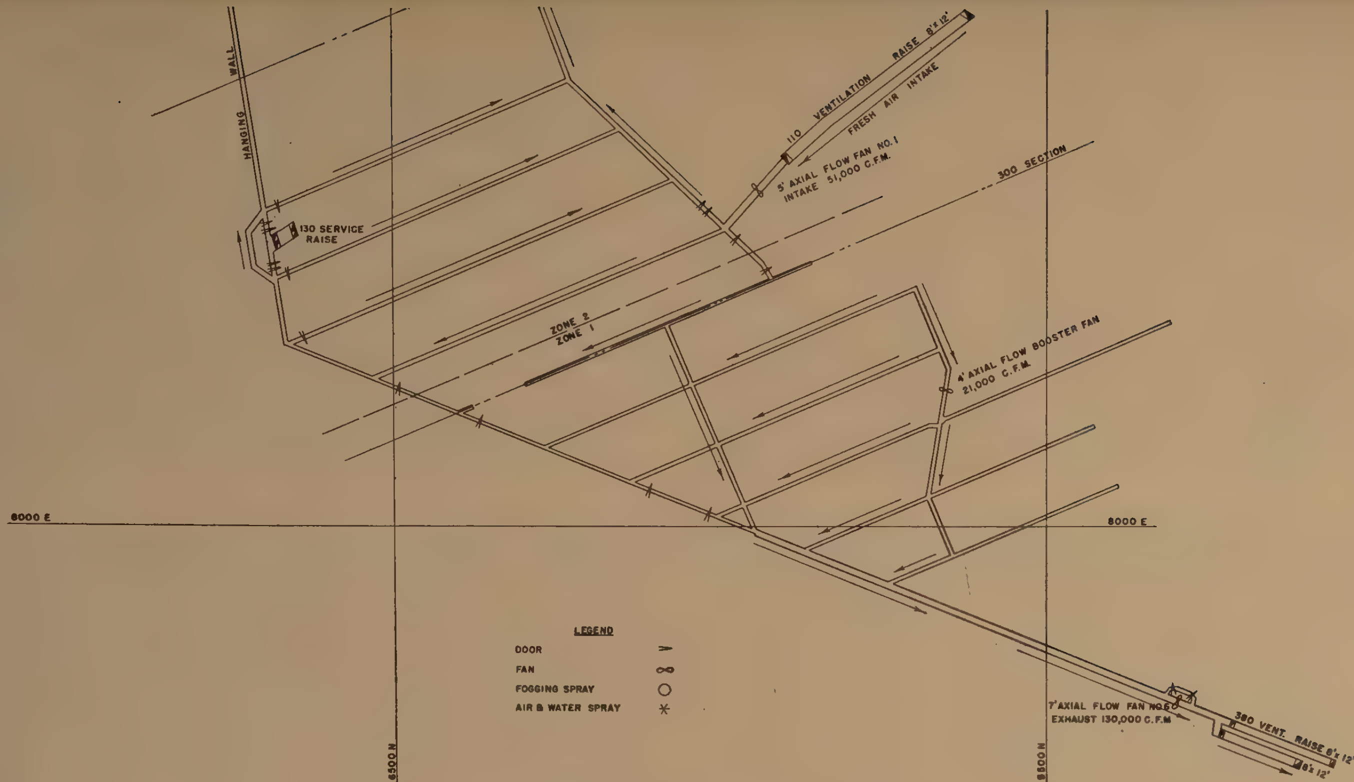

FIG. 3.—GRIZZLY LEVEL.

Zone 1. Air supply to this zone is mostly filtration through the cave area in the overburden, and upcast through open grizzly chutes from the lower main haulage level. Contaminated air from zone 1 is exhausted directly to the surface by the fan at the bottom of 380 ventilation raise.

Zone 2. Fresh air for this zone is drawn down 110 ventilation raise and forced to the hanging-wall drift, from which distribution is effected by the control doors in each grizzly drift. Air volumes entering the grizzly drifts vary from 7000 to 20,000 cu. ft. per min., depending on how many regulating doors are open. Contaminated air from this zone is drawn up through 160 and 220 service raises by two large fans on the uppermost (White) level (elevation 11,940 ft.) and exhausted to the surface.

Sill Plan

On the sill, in the newly developed areas, where production is by the slusher method, footwall and hanging-wall laterals are driven parallel to the footwall and hanging-wall fringe haulage drifts (Fig. 1), and about 50 ft. from them. From these laterals entries are driven to the fringe haulage drifts opposite each loading drift. Control doors of steel and concrete are placed in each of the entries. From these laterals raises are driven to the surface to provide exhaust for contaminated air and intake for fresh air.

Fresh air is drawn from the surface through the hanging-wall raise and is distributed under pressure by an 8-ft. axial-flow fan delivering 100,000 cu. ft. per min. at 1.7 in. pressure along the hanging-wall lateral to all entries. The regulating doors allow a suitable amount to flow to each loading drift.

An axial-flow fan (7-ft.) delivering 176,000 cu. ft. per min. at 2.5 in. pressure, placed in the raise from the footwall lateral, exhausts contaminated air from the entries opposite the loading drifts. After blasting in a loading drift, the control doors in the hanging-wall and footwall entries near this drift are opened wide, allowing a large volume of air to flow through this drift and remove smoke, gas and dust rapidly. Control doors are kept closed except for a short time after blasting.

This arrangement allows a maximum of flexibility and control without the use of doors in the main haulage and loading drift (Fig. 2). Fumes from blasting of stope pillars, in which from 5 to 30 tons of powder are used in one blast, are removed rapidly and are confined to a limited area. Production in other areas is not stopped by these blasts, as formerly was necessary.

In the older parts of the mine, where flow control is necessary on the haulage level, mechanically operated doors are used.

Slusher platforms in the newly developed area are built with a 1 by 10-ft. open space, level with the top of the haulage drift and level with the slusher platform, on the upstream side of the air flow, for better ventilation for the slusher operator. The opening has an apron extending at an angle of about 45° into the fresh-air stream, deflecting fresh air upward and past the operator.

Grizzly Level

The areas that have been developed by the chute and grizzly method are separated into two ventilation zones by isolating one block from the other by bulkheads and doors (Figs. 3 and 4). In zone 1, the oldest zone, one heavy-duty axial-flow 7-ft. fan delivering 130,000 cu. ft. per min. at 1.2 in. pressure exhausts to the surface. Intake air is through caved areas mostly.

In zone 2 a raise (No. 110) for fresh-air intake was driven from the footwall side, and another raise (No. 160) was driven from the footwall area to exhaust contaminated air to the surface. Steel and concrete control doors are placed in the hanging-wall side of all grizzly drifts. The hanging-wall grizzly fringe drift is used as fresh-air inlet and the footwall grizzly drift is used for exhaust of contaminated air.

Fresh air is drawn (by a 5-ft. axial-flow fan delivering 51,000 cu. ft. per min. at 2.2 in. pressure) through 110 drift and forced out to the hanging-wall fringe grizzly drift, and thence northerly to supply all grizzly drifts. Proper distribution is obtained by the regulating doors. Air passing through the grizzly drifts goes to the footwall fringe grizzly drift and is exhausted to the surface by No. 160 and No. 220 fans.

All grizzly doors are kept closed except when blasting has been done. After blasting, the doors are allowed to remain open until the smoke and dust have disappeared. As a large volume of air passes through a door, this is accomplished rapidly.

Temperature Control

Near the intake, at first lateral freezing was a problem. The lowest recorded temperature at the pressure fan was 3°F. To overcome this, and to raise the intake air above freezing, the heat from the cooling system for the compressors is utilized.

The temperature of the water from the cooling system averages 95°. It is forced 1200 ft. through a 4-in. line by a centrifugal pump, at the rate of 200 gal. per min., terminating in a bank of sprays in front of the fan. There are four 1-in. spray lines across the intake raise, set about 3 ft. apart, slope distance, and each 1-in. line has five 5B misting spray nozzles, spraying the comparatively hot water into the air stream. Radiation loss from the passage through the 4-in. line is 7°. The remaining 88° raises 100,000 cu. ft. per min. to a maximum of 41°. Some freezing of the spray from the nozzles occurs during subzero weather, but this is not serious.

For the intake fan servicing the grizzly level, at 51,000 cu. ft. per min., recirculation of exhaust air from an upper level is used. This air is passed through fog sprays, water temperature 40°, and dust counts taken 600 ft. from the sprays have been found satisfactory. Delivery temperature to the grizzly level is 42°.

The circulation of 42,000 cu. ft. per min. from the other intake fan is routed through the compressor room, where it is raised to a temperature of 75° by radiated heat from the compressors and is circulated to other parts of the mine where air tempering is needed.

Recently another fan was installed for the dual purpose of ventilating a sublevel during the period of development and of using the recirculated air to raise the temperature of the older mine workings. This volume of air (30,000 cu. ft. per min.), after passing through three banks of fogging sprays, circulates throughout the entire sublevel and exhausts, at a temperature of 49°, up the main shaft, and is allowed to mix with the colder air of the main level.

Dust Elimination

Actual dust counts have demonstrated that the dust of dust-laden air passing through a curtain of fogging sprays with a moderate velocity is precipitated and entrapped by impingement on the dampened perimeter of the airway. Turbulence caused by the passing of the fog-laden air through timbered passageways aids in elimination of dust. Sprays are placed at points where they are of most value in relation to the fans—mostly close to turbulence caused by the loading of dry ore—and a satisfactory reduction in dust count is thus secured. A sprinkling tank is used for wetting down dusty haulageways with a salt solution where freezing conditions exist and with water only where temperatures are above freezing.

All the newly developed slusher drifts are concreted, and water and air lines are installed in the forms before the concrete is poured. Connections for water and spray heads at points where the draw fingers join the drift are left uncovered at the tops of the drifts. (Described in the *Mining Congress Journal* of January 1941.) During slushing operations the sprays are used to dampen the ore before and while loading.

In the active working zone on the grizzly level, a fogging spray is installed on the intake side at the entrance to each grizzly drift. These sprays keep the entrapped dust on the drift perimeter in a damp condition and retard redistribution of the dust caused by the concussion of blasting. In each grizzly entry is a misting spray acting as a blanket to keep the dust and smoke in the grizzly chamber. Fogging sprays in the main airway saturate the air that is passing through the drifts and keep the wells and bottom damp.

A rigid routine of dust surveys is made weekly throughout the mine, and any abnormal condition is immediately taken

PLAN OF GRIZZLY DRIFT SHOWING ENTRIES AND CHAMBERS

FIG. 4.—ENLARGED VIEW OF TYPICAL LAYOUT OF GRIZZLY CHAMBER ON GRIZZLY LEVEL.
Air and water spray (*) in each grizzly chamber to control dust. A fogging spray (O) to keep the drift perimeter in a damp condition.

care of by rerouting of air currents or the installation of added spray units. The generous use of water sprays, sprinkling and the washing of drift ribs, has been found to be the best preventative of dust.

The type of fogging spray found most effective has a mixing chamber that breaks the water into a fine mist and liberates it as a dense fog. A desirable feature of this spray is that the air and water lines are not directly connected, so that the one with the greater pressure will not back into the other, but escapes through the exit slot. This spray head was adopted after many experiments as being the best suited for the service expected. It was designed at the mine and is assembled in the mine machine shops. Details of the design are shown in Fig. 5.

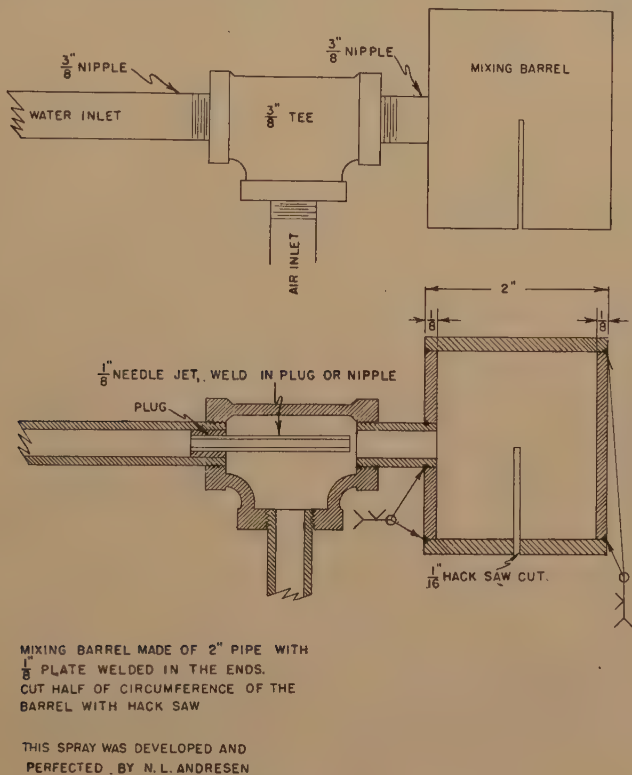

FIG. 5.—FOGGING SPRAY.

After many experiments, this type of spray was found most suited to requirements. It is very efficient, in that it throws a complete blanket of very fine mist. The unequal pressure of air and water (air 100 lb. per sq. in.; water, 254 lb. per sq. in.) was troublesome in other types of sprays tried out.

AUXILIARY VENTILATION

For the ventilation of dead-end drifts, raises and stopes, suction fans of 2500 cu. ft. per min. are used. All are direct drive and powered by 5-hp. motors.

On the 500-ft. sublevel, during exploration drifting, with a drift of 5 by 7 ft., these fans were used. The longest drift was 3600 ft., and three fans were used in the vent line (approximately 1000 ft. apart), all acting as suction units discharging into an airtight compartment of the main shaft and thence

to the surface. Wherever possible the suction system is used in all drifting, as it sucks the drift clear and the smoke and gases from blasting are expelled through

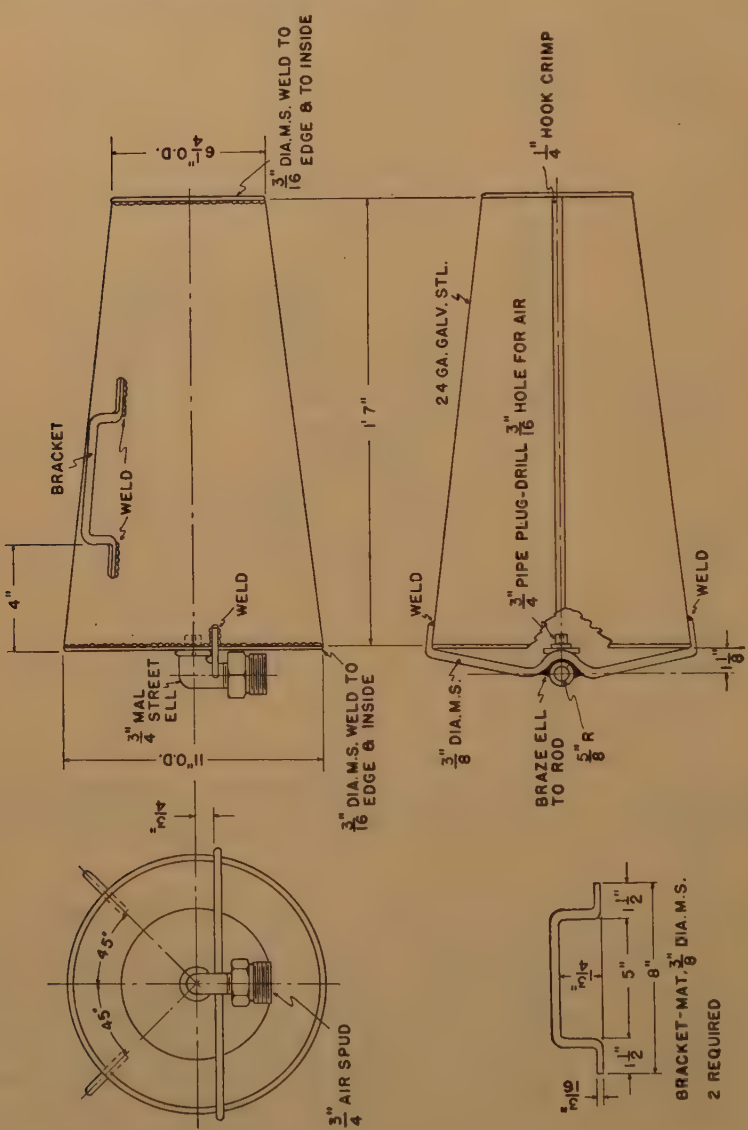

Fig. 6.—Air injector.
Locally called a "Bakooka." It is a modified form of the Venturi air injector, used to boost sluggish circulation in small drifts (5 by 7) and also in short dead-end drifts.

the vent line. The cooling effect of blowing air into a heading is not desirable at Climax. All vent line is 12-in. galvanized pipe, 18 gauge, seams riveted and soldered.

Before blasting is done on the face of a drift or raise, a combined water and air spray is attached to the water and spray lines about 20 ft. from the face; it is turned on before blasting and left on to wet down the muck and kill smoke and dust.

For short dead ends, where it is not practicable to install a fan, a modified form

of Venturi injector is used (Fig. 6). Installed in a 12-in. vent line, in a blowing position with a 100-lb. head of air and a 5/32-in. nozzle, it will deliver approximately 500 cu. ft. per min. through a 100-ft. vent line. These units are used also to boost sluggish air currents in the smaller drifts. The injectors were designed by the mine staff and are assembled in the mine machine shops.

The connections of vent lines are formed of strips of brattice cloth 6 ft. by 6 in., soaked in a thin slurry of cement. A strip is wrapped around a joint and secured with No. 12 wire. The advantage of this type of connection is that the joints can be broken to as much as 2 in. following the rough contour of drifts and on gentle curves, and still be rigid and airtight after the wrapping has set. These connections have been found effective in hundreds of applications; some have been in undisturbed duty for three years.

Auxiliary Dust Control

The conditioned air for the underground bit-grinding room is supplied by a blower fan, which draws air from an adjacent drift through a three-unit double bank of spun-glass air filters.

Exhaust of the milling fines from the hot miller is entrapped in a precipitation tank by the use of water sprays, and the fumes drawn from there and from the electric furnace are exhausted by a suction fan.

In the underground sample room, the crushers, splitter, pulverizer and bucking board are all under hoods connected to a suction fan. There are dampers above each hood for individual regulation. The entrapped dust is passed through a combined air and water spray installed in the 12-in. vent line, exhausting to an upper level. A fogger spray in the short entry leading into the sample room gives the operators at all times comparatively clean conditioned air.

Fan Duties

There are nine permanent axial-flow fan installations circulating 750,500 cu. ft. per min. through the mine workings.

Three of the heavy-duty fans are exhausting units, with a combined capacity of 467,000 cu. ft. per min. Two of these are of the new variable-pitch propeller type. Two heavy-duty fans and one light-duty fan are intake units, delivering 182,900 cu. ft. per min. into the mine. One of the heavy-duty fans is of the variable-pitch propeller type; the remaining three are light-duty low-pressure units and are used as boosters, circulating 100,600 cu. ft. per minute.

All fan installations are underground. Comparative ease of installation accounts in part for the preference shown the axial-flow fans over the more cumbersome centrifugal fans.

Conclusion

The consensus of the Climax mining staff is that the time, effort and expense that have gone into the ventilation program are paying dividends in increased safety of mine operation and production.

Acknowledgment

Thanks are due the Climax mine staff for help in preparation of this article; particularly to Mr. C. J. Abrams, General Superintendent, and Mr. F. S. McNicholas, Assistant General Superintendent.

DISCUSSION

(T. T. Read presiding)

C. M. Smith,* Washington, D. C.—It has been my pleasure to work with Mr. Glanville and other members of the staff of the Climax Molybdenum Co. throughout the development of the ventilation program which Mr. Glanville has so ably described in his paper. While many problems remain to be solved, as the

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paper indicates, the prosecution of a consistent program for ventilating a rapidly growing and highly complex mine has resulted in such improved working conditions as to be a matter of gratification for all concerned.

During the five or six year period, there has been a shift in objectives, which may be summarized in four stages.

In the first stage the aim and necessity was to clear up the visibility. If a positively controlled flow of air through the mine passages is lacking, temperature and humidity conditions are such that fog forms rapidly and causes poor visibility. This condition was interfering with production when the program was started, but it yielded appreciably to the first installation of heavy-duty fans in 1939.

Thereupon the second objective became apparent: to dissipate and remove the powder smoke and fumes produced by secondary blasting. This called for stronger air currents and more definite coursing throughout the mine, a need that was met by installing additional heavy-duty fans and by tying their installation in with a considerable amount of drifting and raising that was done for ventilation purposes only. However, the resulting increase in the flow of air, while serving to sweep out noxious gases, proved to be a mixed blessing, since it led to winter freeze-ups in some working sections, particularly in the 3N. and 3S. blocks. Cold air entered these blocks through the caved ground above them where the cave extends to the surface, and while every effort was made to bulkhead off these openings to prevent excess inflow of air, it was not always possible to prevent freezing and it was sometimes necessary to suspend operations in the slusher drifts involved.

This introduced the third objective, prevention of freezing. This was accomplished by use of hot water sprays and routing of warm air as the author described.

The ventilation program is now in its fourth stage, with the objective of entrapping and exhausting contaminated air without exposing workmen to it. New development plans provide separate passages for return air currents, and much of the present workings is ventilated on the same principle.

The renowned scientist and inventor Steinmetz used to pay tribute to what he termed the "divine discontent" that spurs men on in their search for perfection. It seems that the executives and technical staff of the Climax Molybdenum are thoroughly infected with this virus. While none of the four objectives cited has been attained completely, and perhaps none can ever be brought to perfection, I know that they will always be sought after and that, when their practical limits have been reached, the staff will see a fifth, a sixth and a seventh objective to go after.

What has been accomplished in ventilation at Climax, and what is to be accomplished, are results of a constructive policy, firmly adhered to and progressively administered from top side to bottom boss. As a policy it represents American industrialism at its best; as a practice it represents a genuine contribution to economy and safety of mining. May I commend Climax management on adopting such a policy on ventilation and dust control, and Mr. Glanville with his associates at the mine on the way they are carrying it out.

Progress in Air Conditioning for the Ventilation of the Butte Mines

By A. S. Richardson,* Member A.I.M.E.

(New York Meeting, February 1941)

Air conditioning, or air cooling, for the improvement of ventilation in the Butte mines has come into increasing use during the past 10 years. In part, the methods practiced have been described in previous publications of the Institute, but some improvements on earlier methods have been made and a number of new features have been developed. This paper is presented as a record of progress made, and for the sake of completeness describes all general features of the work.

The need for air conditioning in the ventilation of mines in the district arises from the fact that the temperature of the ground and mine water has shown a very great increase with increase in mining depth, and has now reached a maximum of slightly over 130°F. The most difficult part of the ventilation problem is the maintenance of temperature and humidity of a satisfactory standard in the working places, especially on the lower levels.

Although ordinary ventilation is adequate to maintain good conditions in most of the working places it is subject to a number of limitations at great depth. Most important among these limitations is that with increase in depth the air becomes heated before it reaches the working places, both by compression from increase in atmospheric pressure and by heat transferred to it from rock at high temperature, so that its capacity for lowering temperatures is greatly reduced before it reaches the operating zone. Increasing depth of mining also increases the distance traveled by the air in its passage through the mine, and this causes a corresponding increase in resistance to air flow, so that there is greater difficulty in maintaining even the same volume in circulation. Thus, in the face of a demand that the ventilating system have a very much greater capacity to remove heat from the mine, the tendency is for that capacity to diminish. The only evident alternatives were (1) the opening of additional air shafts to increase the volume of air in circulation, (2) an extension of the existing system that would be subject to the same limitations, or (3) the adoption of an entirely different method such as air cooling.

Fortunately for an economic solution of the problem, the mean annual temperature of the surface atmosphere at Butte is about 40°F., with very low relative humidity in summer, so that there is unlimited natural cooling power at a temperature range adaptable to air-conditioning methods.

Methods

Two similar methods of air conditioning are used, as necessitated by service requirements and dependent upon the equipment available. In one, which is the more important and has the greater cooling capacity, water is circulated in a closed circuit, and, alternately, first absorbs heat from warm mine air and then dissipates it in the surface atmosphere by cooling at a cooling tower. The second

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general method is to use cold water, derived either from a source within the mine or from the fresh-water supply system, to absorb heat from the mine air, and then to turn it to waste in the mine drainage system.

The success of both methods depends upon the fact that, because of the greater density and specific heat of water, a unit volume of water will absorb much more heat than would be absorbed by the same volume of air for the same increase in temperature. A relatively small pipe line, therefore, will carry enough water to remove as much heat from a mine as would be carried by a volume of air for which a large shaft would be required. Also, the water is not subjected to any increase in temperature due to increase of pressure, and by insulation of the pipe lines the heat transferred to it from surrounding warm air may be more closely controlled.

The equipment first used when circulating water in closed circuit is illustrated diagrammatically by Fig. 1. The water (brine in winter) is cooled at the cooling tower and then flows through a pipe line to an underground air-conditioning plant where it absorbs heat from the mine air; it returns through another pipe to the cooling tower. The underground plant is composed, essentially, of a number of coils of small-diameter pipe, which form part of the whole pipe system, so that the hydrostatic head is in balance, and power, furnished by a pump on the return pipe line, is required only to overcome frictional resistance to water flow.

Cooling of water at the surface cooling tower is naturally more difficult during the summer than in winter, and it has been necessary to use a special method of utilizing the exceptionally dry climatic conditions.

In the Butte district, during the summer, the sensible temperature seldom exceeds 90°F. for more than a few hours per day, or during more than two or three weeks time. Under such a condition the wet bulb usually will be about 60°F., so that the dew point is about 39°F. Frequently, the dew point is below the freezing point, even during the month of July.

Since it is not possible to cool water below the wet-bulb, or evaporation temperature, of air brought into contact with water at a cooling tower, ordinary counter-current circulation would not give satisfactory results during the summer, especially since it is not practically possible to realize theoretical limits. However, a system was developed that makes it, theoretically, possible to cool the water to the dew-point temperature of the air, instead of the wet-bulb temperature, and this satisfactorily solved the problem.

Fig. 1 gives the best explanation of this system. This shows that for summer operation there are two separate, closed-circuit systems of water circulation at the cooling tower. In one of them warm water returning from the mine is delivered to the top of the tower, and after being cooled falls into the sump; from which it passes through the mine pipe lines and in return back to the top of the tower. In the other closed circuit, water is also drawn from the sump by a pump and forced through a heat absorber in the air duct between the fan and cooling tower, and then back to the top of the tower. Water from both circuits, of course, mixes freely in passage downward through the tower and into the sump.

The function of the heat absorber in the duct between fan and tower is to precool the air, before it reaches the tower, down to a wet-bulb temperature of, say, 47°. Since this heat absorber is similar to an automobile radiator, there is no contact between air and water; therefore the moisture content of the air (and consequently its dew-point temperature) is not changed, although its sensible and wet-bulb temperatures are lowered.

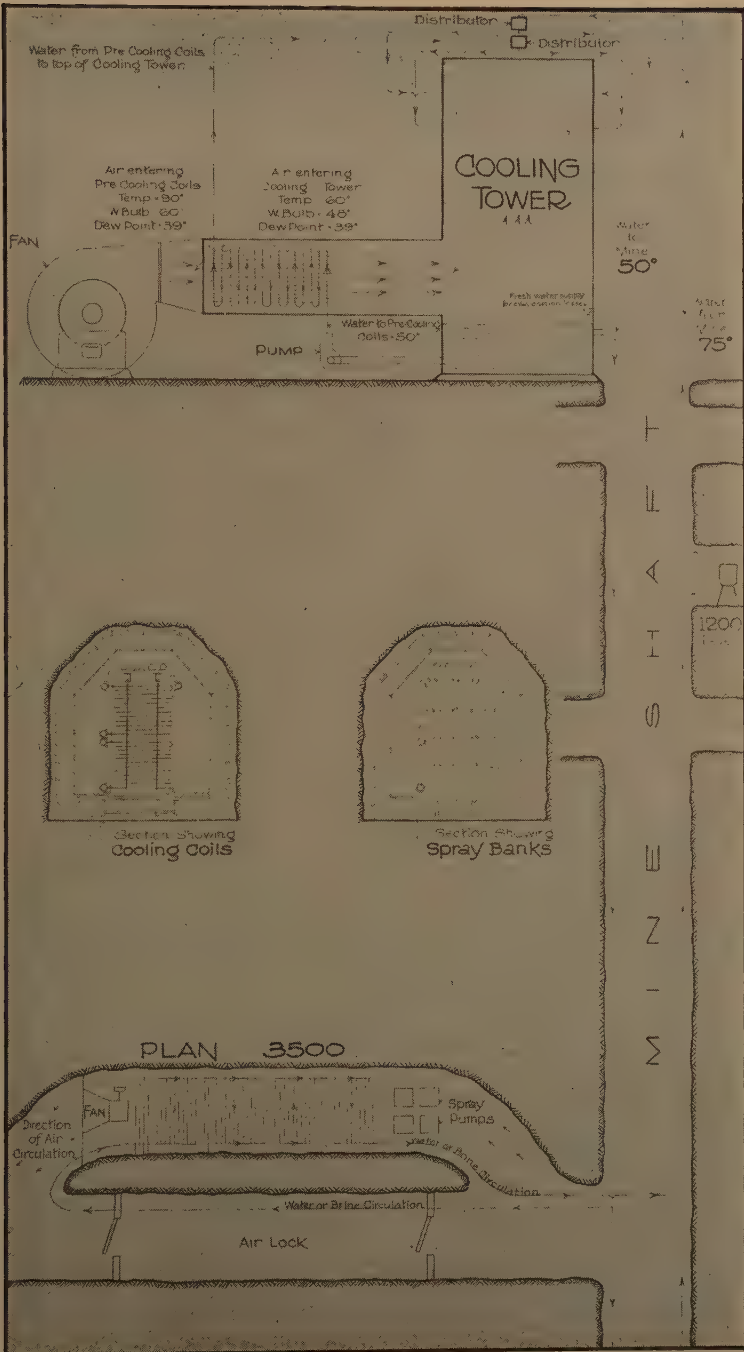

FIG. 1.—GENERAL PLAN OF AIR-CONDITIONING PLANT.

To understand how this result is produced, assume that air at a temperature of 90°F. (60°F. wet bulb) is delivered to the cooling tower, and that water is thereby cooled to the theoretical limit of 60°F. If this water is pumped through the heat absorber, in countercurrent heat exchange with the air, its temperature theoretically may be raised to 90°F., while the sensible temperature of the air will be lowered to 60°F. Since heat has been removed by the water from the air its wet-bulb temperature, which is proportional to the heat condition of the air, will be lowered to approximately 47°F. After this heat exchange air would be delivered to the bottom of the tower with a sensible temperature of 60°F. (wet-bulb temperature 47°F). However, with air at a wet-bulb temperature of 47°F. entering the bottom of the tower, the water theoretically may be cooled to 47°F. When this colder water is pumped through the precooling heat absorber the wet-bulb temperature of the air would be still further lowered. The process is evidently regenerative until the dew-point temperature of the air is reached, or when sensible, wet-bulb, and dew-point temperatures become the same.

In this cycle of operations, heat is withdrawn from the air entering the tower by the precooling heat absorbers, and returned to it in the cooling tower. Such an operation may be regarded doubtfully because it involves merely the transfer of heat out of and back into the air, so that no real advantage seems to accrue. However, in ascending the tower this precooled air not only takes up from the countercurrent water all the heat that was abstracted from it in the precooler but a considerable amount in addition, by evaporating the water. That which does not evaporate reaches the bottom at the wet-bulb temperature of the incoming air. The advantage of the precooling is to bring the wet-bulb temperature of the air, at the point where it encounters the water in direct contact, down to the desired temperature for the cooled water, which otherwise could not be attained. Thus the precooling of the air permits the cooling of the water to a lower temperature than could be attained through the use of uncooled air.

Underground Plants

Designs of the underground plants show some changes at units successively installed. From early experimental work, it was known that with the lowering of the temperature of warm, humid, mine air, and consequent condensation of water vapor carried by it, a large part of the dust carried in the air would be precipitated by the condensation of the water vapor upon the dust particles as nuclei. The cumulative effect of such a cleaning of the continuous flow of a large volume of air was the deposition of a film of dust on all surfaces of the air-conditioning equipment. This naturally tended to reduce the transfer of heat from air to water, and the possibility of having to clean the heat-transfer surfaces has been an important factor in planning all designs.

The first high-pressure plant that was installed on the 3500-ft. level of the Mountain Con mine was, accordingly, designed to facilitate such cleaning. It is composed of four groups of parallel-flow coils of ¾-in. diameter pipe, which are connected in series and through which cold water is circulated in a direction countercurrent to that of the air. For each of the groups of parallel-flow pipe coils there is a separate system of water sprays in which the water, after having been sprayed into the air and in contact with the pipe coils, falls into a sump from which it is drawn by a pump and again forced through the sprays in closed-circuit circulation. The action of these sprays is to wash the air and to transfer heat from it to the water inside the pipe coils. Between the four groups of pipe coils there are

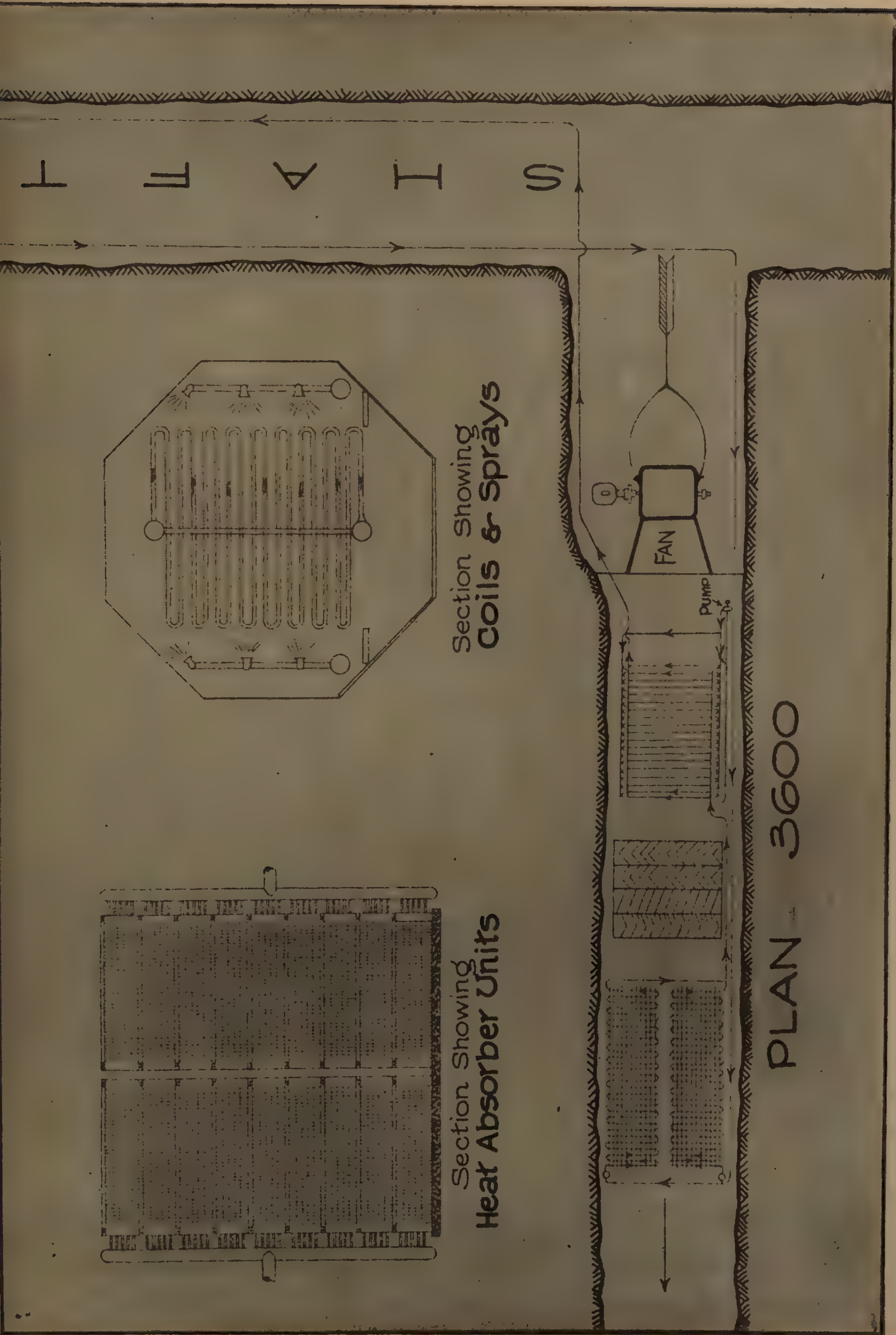

FIG. 2.—AIR-CONDITIONING PLANT ON 3600-FOOT LEVEL.

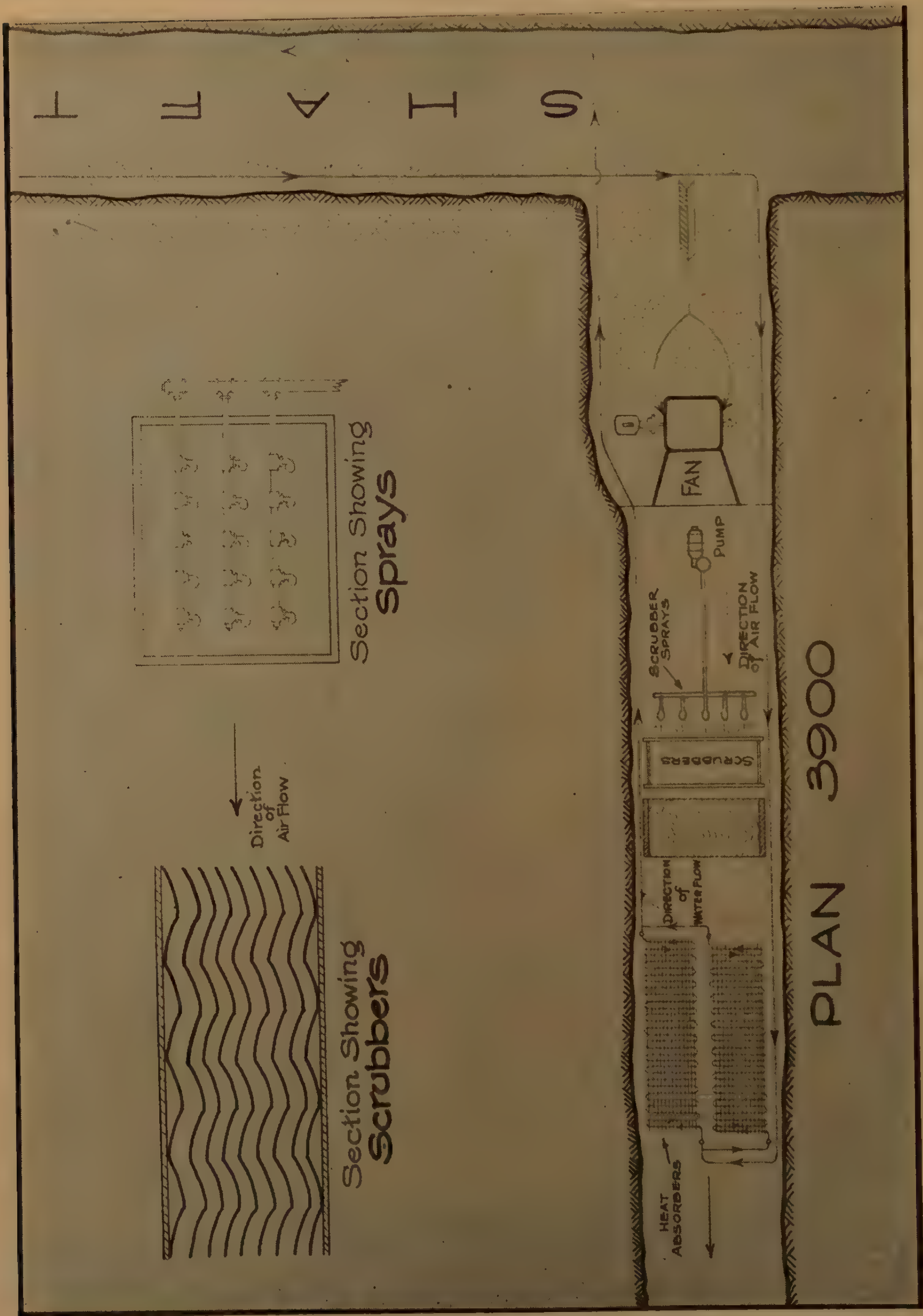

FIG. 3.—AIR-CONDITIONING PLANT ON 3900-FOOT LEVEL.

baffle plates, or eliminators, to prevent the spray water from being blown downstream with the air.

From the operation of this plant it was learned that about 90 per cent of the dust, by weight, was eliminated at the first set of sprays, so that the effect of dust precipitation on equipment in the other three sets of coils was relatively slight. The space occupied by this plant was very large, the length of the chamber required for the pipe coils being about 65 ft. The ground is so bad that it is extremely difficult and expensive to maintain a large opening, so that it was highly desirable to reduce the size of the plant.

With that information and purpose in mind the design of the next unit, on the 3600-ft. level, was changed as shown in Fig. 2. At that plant, only one stage of ¾-in. pipe coils and sprays is used, and this serves to remove the dust by washing and slightly cooling the air as it enters the plant. The major part of the cooling effect is then accomplished by passing the air over an extended surface type of heat absorber, which occupies only a small part of the space required for the three stages of pipe coils and sprays that it replaces. Some changes in the arrangement of pipe coils and sprays are shown in this figure, but there is no essential change in operating characteristics. The direction of water flow is counter current to that of the air, first through the heat absorber and last through the pipe coils in the spray chamber.

Following the installation of this plant, some experimental work was done to see whether fine dust might not be eliminated from mine air by means of baffle plates. It is believed that most past experience has indicated that satisfactory results could not be achieved by this method. However, the fact that the impinger dust-sampling instrument depends for its success upon the impingement of the dust-carrying stream of air upon a wetted surface seemed to indicate that there might be some possibility of getting the desired results if the important factors of design were properly put into use. The essential, or important, features of impinger design and operation contributing to its success seemed to be the impingement of a small stream of air at such high velocity against the bottom of the impinger flask that the momentum of the dust particles carried them into direct contact with this wetted surface.

Taking the same line of attack, a set of closely spaced baffle plates with closed-circuit spray system was designed as shown in Fig. 3. Each plate is formed into arcs of circles, alternately reversed as to direction of inclination to the main line of flow, and the space between the baffle plates was made small, from ⅜ to ¾ in. Arrangement of the individual plates into alternately reversed arcs of circles was impelled by a desire to utilize centrifugal force, and also to cause the stream of air to be sharply impinged against each succeeding arc. The purpose in close-spacing the plates was to cut the volume of air into narrow streams, so as to reduce to a minimum the distance through which the momentum of the dust particles would need to cause them to be deflected in order that they might be impinged against the baffles.

In testing this device it was found, as previously expected, that for anything like successful results it was necessary to pass the air through it at higher velocities than have commonly been used in air-conditioning work. The orifice velocity for the standard impinger is approximately 22,000 lin. ft. per min., while for the commercial midget impinger it is about 12,000 lin. ft. per min. With these baffles really good results were not obtained until a velocity of about 2000 ft. per min. was reached. However, at that velocity the elimination of dust was about equal to that obtained by the dehumidifying effects developed at the air-conditioning plants already in

operation, so that it was then practicable to eliminate the chamber containing the pipe coils and sprays and further reduce the size of the plant. No exhaustive study of this baffle-plate dust eliminator was made because a long time would have been necessary for the work and the practical purpose had been gained.

The next air-conditioning plant installed on the 3900-ft. level was then designed to utilize this dust eliminator, followed by extended surface heat absorbers, but without any long chamber containing pipe coils and sprays. All plates in the dust scrubber, or eliminator, lie in a nearly horizontal plane, because it was found that when such plates are set in a vertical position the spray water entering with the air immediately begins to drain downward when impacted against the plates, and this leaves a great part of the plate surface without any washing action to remove accumulated dust. The baffle plates used to eliminate water discharged from the dust scrubbers and return it to the spray system are set vertically.

Construction of the Piping System and Underground Plants

The main pipe lines that are carried down the mine shaft are 8⅝ in. o.d. weldless tubing, grade B steel, with wall thickness necessary to sustain the hydraulic pressure with a factor of safety of at least 4 at all points. The individual pieces of tubing received from the mill were welded together at the surface to make single pieces about 75 ft. long, with a flanged coupling on each end. After welding they were given a coating of insulating material (1 in. of rock wool) protected by a weatherproofing sheet of asbestos and asphalt bound in place by copper wires. The welded and insulated pieces of tubing were then lowered down the shaft by a worm-geared hoist and the line was put into place, commencing at the bottom and building upward. Support is provided at vertical intervals of about 500 ft. by suspension from steel bearers that are carried into solid ground independent of shaft timber. Above each bearer there are U-shaped pieces to take up expansion and contraction due to temperature changes, and these have also proved valuable in compensating for changes in the distances between bearers caused by ground settlement. The bottom of the shaft line is on the 3500-ft. level, and below that elevation the further extension to depth is carried through raises in country rock nearer the vein, which were driven especially for this purpose.

The pipe lines on the mine levels that serve the different underground plants are 8⅝, 6⅝, or 4⅝ in. outside diameter as necessary for the volume of water to be passed through them. When it is possible that the pipe lines may be moved from one level to another, the wall thickness is made heavy enough to withstand the hydraulic pressure exerted by brine having a head of 5000 ft. and a specific gravity of 1.2. The individual pieces of pipe taken underground are usually about 15 ft. long, and three pieces are welded together to form one piece having a flange coupling on each end. A number of special pieces for bends and shorter lengths are required. Support is provided by hangers from timber or ground, depending on mine conditions.

The main circulation pump is placed on the return line at the 1200-ft. level. It is a four-stage centrifugal pump, and is direct-connected to a 3450-r.p.m. motor. The rated capacity is 1000 gal. per minute against a head of 750 ft. The glands are lubricated by fresh water from the main fresh-water supply system, and this water makes up for part of the evaporation losses.

The spray systems used at the underground plants include no novel features of design. Where heat is to be transferred it is the practice to circulate about one pound of water per pound of air. Where

the water is to be used only to wash dust from scrubber surfaces, a much smaller quantity may be used.

The heat absorbers used at the underground plants were first made up of pieces of ⅝-in., heavy-wall, copper tubing, having 42 square copper fins, each 2 by 2 in., per linear foot of tubing. The bond between fins and tubing was then obtained by expanding the dead-soft copper tubing into the medium-hard copper fins by means of a mandrel. Individual small fins have been eliminated in later designs, and they have been replaced by copper sheets running the full length of the heat absorbers, from 4 to 10 in. high. These medium-hard copper sheets are drilled to receive the dead-soft copper tubes, which are expanded into them by hydraulic pressure. All heat absorbers built for underground service are also made heavy enough to withstand the pressure of brine at a 5000-ft. head and a specific gravity of 1.2, with a factor of safety of at least 4. On account of corrosion, it has been found necessary to use copper for all equipment such as dust scrubbers, water eliminators, etc., used at these plants.

Operating Results

It was planned to have four underground plants and four cooling towers at the surface, with the idea that only two of the towers would be used for about eight months of the year, the other two being brought into service only when precooling the air to the towers was necessary, or during the summer months. After construction of the first two towers, it was found that by increasing the volumes of both water and air passed through them the final temperature of the water returned to the mine would be only a few degrees higher than originally planned. Since such an increase of only a few degrees in the temperature would be of practical importance for only a few weeks during the summer, it was decided therefore not to construct any more towers, but to make some changes to fan equipment and provide each fan with two motors, direct connected at opposite ends of the fan shaft. One of these motors is a relatively high-speed, high-power motor that is used during the summer, and the other is a slow-speed, low-power motor that is used during about eight months of the year. The two towers that have been built are identical as regards essential features of design, and results realized in actual operation are given in Table 1. The first column shows results with No. 1 tower during summer months when operating under conditions for which it was designed. The second shows results obtained with No. 2 tower when operating during the summer at increased fan speed, and with increased water and air flow. The other columns show

TABLE 1.—*Operation of Towers*

Air and Water	No. 1 Tower, July 1, 1937	No. 2 Tower, July 12, 1940	No. 1 Tower, Dec. 10, 1940	No. 2 Tower, Dec. 10, 1940
Volume of air passing through tower, cu. ft. per min.	83,000	129,000	87,000	75,000
Static, or resistance, pressure in inches of water	2.90	6.70		
Static air horsepower	38	136		
Volume of water through precool coils, gal. per min.	130	230	None	None
Volume of water through mine circuit, gal. per min.	200	400	390	360
Temperature of air from fan to precool coils, deg. F.	83	92	24	23.5
Wet-bulb temperature of air from fan to precool coils, deg. F.	54	58.5		
Temperature of air to bottom of tower, deg. F.	57	65.5	24	23.5
Web-bulb temperature of air to bottom of tower, deg. F.	44	47.5		
Temperature of air from top of tower, deg. F.	68.4	70.0	56	58
Wet-bulb temperature of air from top of tower, deg. F.	68.2	69.6	55.8	57.4
Temperature of water from mine to tower, deg. F.	76	74	61.5	61.5
Temperature of water from precool coils, deg. F.	72.5	80	None	None
Temperature of water to mine and precool coils, deg. F.	46	52	32	32

both towers under winter conditions of operation with slow-speed motors running.

During the month of July, 1937, with a mine flow of about 200 gal. per minute through one tower, for which it was designed, the average temperature of the water returned to the mine was 48.2°F. During the month of July, 1940, with a mine flow of about 750 gal. per min., and one fan with increased air flow, the average temperature of the water returned to the mine was 52°F.

With the installation of more underground units, the flow of water will be increased to about 1000 gal. per min., or the full capacity of the pump and pipe lines. At the present time the fan on No. 1 tower is an old belt-driven, single-inlet, forward-tipped blade centrifugal that was put on for temporary use only while estimates as to the performance of the tower were verified. It is about 25 years old, and is not as efficient as more modern fans. A new fan will be installed on No. 1 tower when the flow of water is further increased, and it will be equipped with two motors, for summer and winter conditions, one motor on each end of the fan shaft. Under this arrangement the power consumption during the summer months will be high, but the building of more cooling towers will not be necessary.

Temperature rise in water, or brine, between cooling tower and underground plants was estimated at 5°F. Actually, it has run from 6° to 10°F. at different times of the year. It is believed that this is partly due to ineffectiveness of insulation on mine levels. Where the pipe lines pass through warm, humid air, there is an infiltration of air into the insulation, and contact with the cold pipe surfaces causes a condensation of water that partly fills many of the voids in the rock wool. Insulation on the shaft lines is dry, except near the bottom. On some of the lines that have been installed recently on the mine levels, no insulation has been attempted, but the pipe is protected against corrosion on the outside by a heavy coating of biturine, which probably has some insulating value.

Prevention of corrosion of the inside surfaces of the pipe lines, and keeping the water, or brine, clean requires careful attention. About 15 gal. of water per minute are continuously added to the system from the fresh-water supply, to compensate for evaporation losses, and there is an accumulation of salts in solution from this source. Also the atmospheric dust in the air forced through the cooling towers enters the water circulation system and adds some fine solid material. As an anti-freeze, or brine, it has been found best to use sodium chloride and sodium phosphate, with the addition of corrosion inhibitors such as sodium dichromate, and to keep the pH up at about 9 with sodium hydrate. In summer, water is used with the same inhibitors in lower concentrations, and the pH is kept at about 8. Screens and pipe strainers are used to eliminate solids continuously, both at the surface cooling tower and at the underground plants. Thin strips of steel, copper, and brass, are kept in the sump of the surface cooling tower, and are weighed periodically to check losses of metal. Inspections of the pipe lines are made at intervals of about three months, by removing small sections of the lines. Since there is no apparent change in the condition of the lines that have been in use for five years, they should have an indefinitely long life.

Operation of the underground plants, to obtain maximum benefit for the mine as a whole, is based upon recirculation of a large part of the air passing through the plants and, to a large extent, this is made necessary by the limitations of the existing ventilation system. The total volume of fresh air circulating through the lower levels of the mine, or the section directly affected by the operation of these plants, is only slightly more than 100,000 cu. ft. per min., but the total volume of air

passing through the plants has at times been more than 150,000 cu. ft. per min., and when plants now under construction have been installed it will be more than 250,000 cu. ft. per minute.

One plan that has been followed is to allow the re-conditioned air from the plant to flow through the workings to be cooled, after which about 25 per cent of its volume is discharged into the outlet air course. The remaining 75 per cent of the volume is increased to its original volume by the addition of fresh air from the main circulation system and returned to the plant. The addition of this fresh air maintains the oxygen content of the air at its original standard, and plants have been operated successfully with 90 per cent of the air volume circulating in closed circuit. By this means, the rate of air circulation through a section of a mine may be indefinitely increased, subject only to the limitations of mine resistance to air flow, with marked benefit as to cooling power resulting from higher air velocities, and air formerly used to ventilate the section may be released for the benefit of other sections of the mine not directly affected by the operation of the air-conditioning plants.

The volume of air circulated through the plant installed on the 3500-ft. level was at first close to 65,000 cu. ft. per min., but after nearly five years of operation a large part of the ore in the area serviced by the plant has been mined out. The plant probably will be used at this place for about two more years, but the volume of air circulated through it has been reduced to 35,000 cu. ft. per minute, and the flow of water has been proportionately reduced.

The volume of air circulated through the plant on the 3600-ft. level has varied from 50,000 to 78,000 cu. ft. per min. at different times. This unit will soon have been in service for four years. Another unit is now being installed farther west on the same level, and after it is in operation the plant now on the 3600-ft. level will be moved to the 4100-ft. level.

Volume of air passed through the plant on the 3900-ft. level has varied from 50,000 to 63,000 cu. ft. per min. It has been in service since March 1940, and will remain in use at the present location for some time to come.

Another plant is being installed on the 3800-ft. level in a more easterly section of the mine, which will circulate a larger

TABLE 2.—*Operating Data*

Air and Water	3500-ft. Plant, Aug. 24, 1936	3600-ft. Plant, Oct. 16, 1937	3600-ft. Plant, Jan. 12, 1940	3900-ft. Plant, Apr. 11, 1940	3900-ft. Plant, Sept. 12, 1940
Air volume, cu. ft. per min.	59,000	50,000	76,000	63,000	50,000
Intake air: temperature, deg. F.	85.5	86.0	88.0	82.0	90.0
Wet bulb, deg. F.	84.5	85.5	84.0	79.5	83.5
Humidity, per cent	96	98	85	90	77
Grains per cu. ft.	12.41	12.86	11.85	10.46	11.31
Discharge air: temperature, deg. F.	72	68	61	61	67
Wet bulb, deg. F.	72	68	61	61	67
Humidity, per cent	100	100	100	100	100
Grains per cu. ft.	8.24	7.48	5.94	5.94	7.24
Water, or brine: gal. per min.	240	200	400	370	290
Intake temperature	54	50	46	47	61
Discharge temperature	80	84	76	68	81.5
Water removed from air, gal. per min.	4.46	4.8	7.7	4.9	3.5
Cooling effect, B.t.u. per min.:					
Sensible heat	15,000	15,800	32,500	21,000	18,000
Vapor condensation	37,000	40,200	68,000	44,000	32,000
Total heat	52,000	56,000	100,500	65,000	50,000
Equivalent tons of ice melted per day	260	280	502	325	250

volume of air, about 100,000 cu. ft. per min., without as much lowering of the temperature as occurs at the other plants. When all these plants are in operation, the full capacity of the pump and piping system will have been reached.

The general data for the different plants when operating on the dates mentioned are given in Table 2.

Dust Elimination

Elimination of dust from mine air effected by these plants was determined by taking samples simultaneously at the inlet and discharge of the plant with standard impingers. Since selective elimination of coarser particles would naturally be expected, some determinations of particle-size distribution were also made by taking samples with an Owens jet dust sampler. These samples were not taken simultaneously, but they were taken at the two places within an interval of only a few minutes. But little variation in average size was indicated, except that some increase in average size occurred during blasting at both intake and discharge. The geometric mean diameter varied from 0.30 to 0.35 microns at the intake, and from 0.25 to 0.40 microns at the discharge. Standard geometric deviation varied from 2.18 to 2.96 at the intake, and from 1.78 to 2.36 at the discharge. Microscopic work was done by microprojection at a magnification of 10,000 diameters. Thus, it does not seem that there was any selectivity as to size in the elimination of dust.

The results of simultaneous impinger sampling that are directly comparable are given in Table 3. For the sake of getting reliable comparisons it was necessary to take samples during the blasting period and immediately afterward, because dust concentrations are usually so low during the regular working period that it is difficult to make satisfactory determinations. Sampling time was varied from five minutes to one hour in order to get a large enough number of particles for accurate counting.

Table 3.—*Results of Simultaneous Impinger Samples*

Plant	Number Concentration[b]		Elimination, Per Cent
	Plant Intake	Plant Discharge	
3500-ft. plant: blasting period[a]	166.00	62.40	62.41
Blasting period	10.36	4.18	59.65
Regular work period	0.91	0.50	45.50
Average			55.85
3600-ft. plant: blasting period	45.55	24.67	45.84
Blasting period	40.91	12.26	70.03
Regular work period	0.62	0.36	41.93
Average			51.93
3900-ft. plant: blasting period	84.57	18.25	78.42
Blasting period	12.72	5.46	57.00
Regular work period	0.83	0.21	74.69
Average			70.00

[a] Blasting period is after the regular shift.
[b] Figures for number concentration are in millions of particles per cubic foot.

In experimental work on the dust eliminator used at the 3900-ft. plant the results obtained were not quite as good as those given in Table 3; for the same air velocity, they were about 50 per cent. The distinction as to conditions was that in the experimental work the spray water, being in closed-circuit circulation, naturally assumes a temperature equal to the wet-bulb of the incoming air, whereas at the 3900-ft. plant the spray water was continually cooled by the addition of colder water condensed out of the air by the heat absorbers. Thus, a combination of dehumidification and impingement accounted for these better results.

Improvement of Conditions in Working Places

Direct comparisons of conditions in working places directly affected by the operation of these plants is possible only to a limited extent. Excavation required for the installation of the plants, and other near-by development work, is now done

at most places with improvement of ventilation conditions by small air-conditioning plants that will be described later, and no more work is done in this section of the mine than is absolutely necessary until after the plant is in operation. With better ventilation conditions then established, further development and stoping goes ahead with a great increase in the number of places that are active, but this affords no basis for comparison as to what conditions would have been if the air conditioning plants had not been installed.

When the 3500-ft. plant was installed the area directly affected had been cooled to some extent by another plant that had a capacity of about 50 per cent of that of the newer plant. The maximum improvement in any one place was from 90° to 78°F., and the average reduction in 14 working places was from a temperature of 86°F. (83°F. wet-bulb) to 80.5°F. (78°F. wet-bulb).

When the 3600-ft. plant was installed the maximum improvement in any one place was from 94°F. (92°F. wet-bulb) to 62°F. (62°F. wet-bulb). The general average temperature and wet bulb in 14 working places was reduced from 90.4°F. (88.2°F. wet-bulb) to 76.9°F. and 74.4°F. wet-bulb. An improvement in average conditions for all of the 103 working places in the mine was from 87.1°F. (83.7°F. wet-bulb) to 78.7°F. (75.3°F. wet-bulb). This general improvement was largely made possible by the closed-circuit circulation of air in the air-conditioned areas that liberated more air for the ventilation of other sections of the mine. About five months was required to effect the change, and since it occurred during the winter it is estimated that a lowering in average wet-bulb temperature of somewhat over 2°F. must be attributed to a natural seasonal change.

When the 3900-ft. plant was put into operation the maximum improvement in any one place was from temperature 88°F. (85°F. wet-bulb) to temperature 70°F. (69°F. wet-bulb). The average for eight places was from temperature 88.1°F. (86°F. wet-bulb) to temperature 76.1°F. (73.1°F. wet-bulb). Very little work had been done in the area, and the flow of air was restricted because only one raise was holed on the discharge side.

Possibly the best general measure of accomplishment is registered by the difference between ground temperature and average temperature of mine air under different conditions. The highest ground temperature noted on the 3200-ft. level in freshly opened ground, before the operation of any air-conditioning plants, was 97.5°, and the average condition for all the working places in the mine was temperature 85.5°F. (wet-bulb, 82°F). On the 4100-ft. level, the highest ground temperature in freshly opened ground was 118°F. and with three air-conditioning plants in operation, the average mine condition, for the same month of the year as before, was temperature 82.7°F. (wet-bulb, 80.0°F). The difference between maximum ground temperature and average temperature of the mine air was, in the first case 12°F., and in the latter 35.3°F.

Low-pressure Plants

Although the plants that thus far have been described are the only plants now operating for which closed-circuit circulation of water, or brine, is provided, a much larger number of smaller plants have been used, which are operated with water that is discharged to waste in the mine drainage after it has passed through the plants. They may be classed as low-pressure plants and in general they have served three main purposes: (1) as experimental plants to obtain data for the design of the high-pressure, closed-circuit circulation plants; (2) to make use of relatively cold water entering the mine on upper levels in cooling lower and warmer sections of the mine before turning this water into

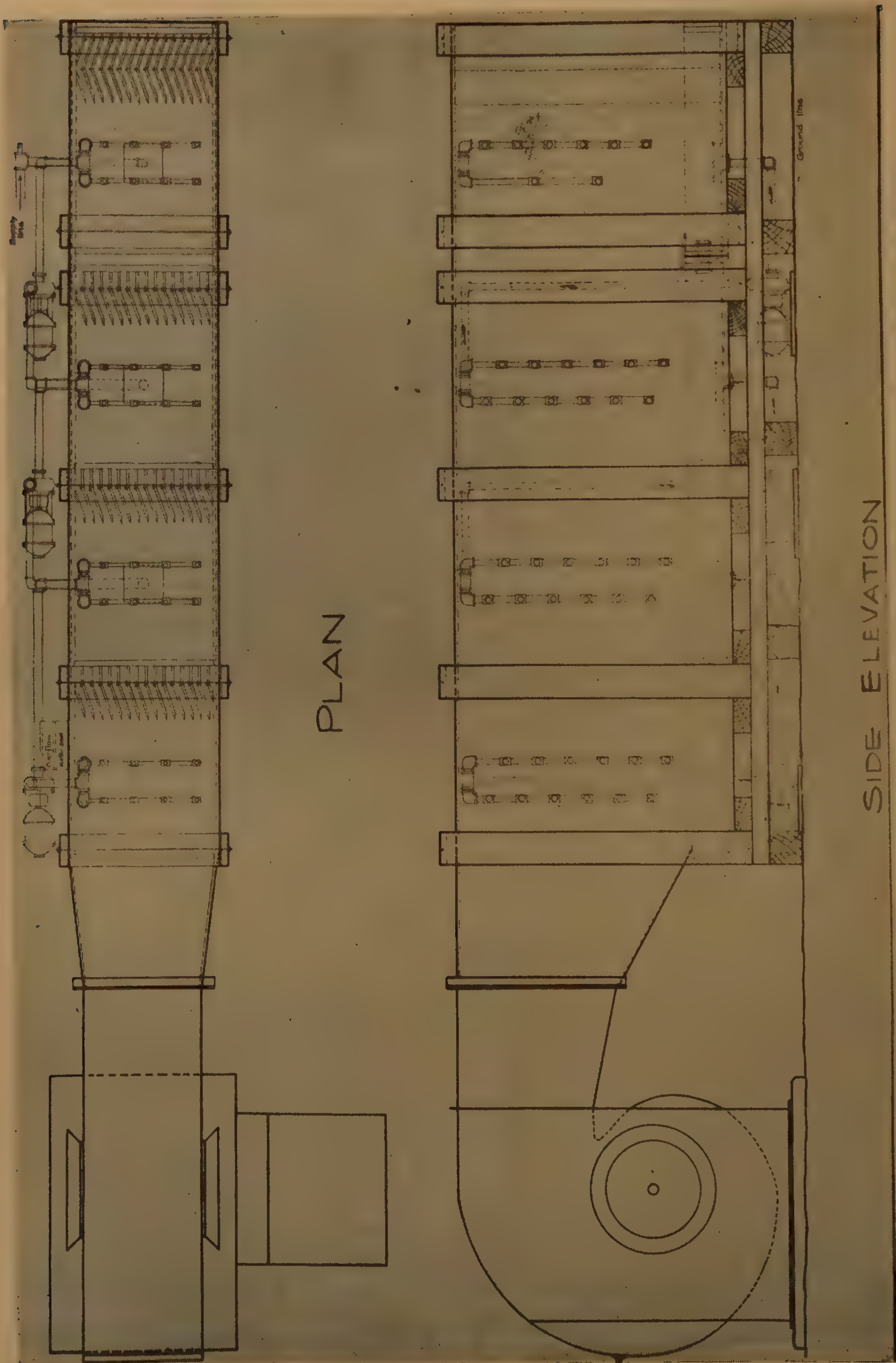

FIG. 4.—FOUR-STAGE AIR-CONDITIONING PLANT, WATER-SPRAY SYSTEM. CAPACITY 30,000 CUBIC FEET PER MINUTE.

the drainage system and pumping it to the surface; (3) in temporary service using cold water from the mine fresh-water supply to improve conditions in dead-end development headings; or larger installations used to improve conditions in sections of mines not yet equipped for closed-circuit circulation plants.

The plants used for experimental purposes were closely similar to the 3500-ft. level plant that has been described, and need no further mention.

On the 1700-ft. level of the Steward mine, there is a flow of about 90 gal. of water per minute from old workings, long closed, varying in temperature from about 47°F. during the winter to 52°F. during the summer. It carries a quantity of mineral salts in solution, is acidic (pH 3.8) and is corrosive enough to destroy an iron pipe in about three weeks.

Above the 3200-ft. level some stopes were exceptionally warm, and conditions could not be much improved by ordinary ventilation methods without detriment to other sections of the mine. Utilizing the flow of cold water on the 1700-ft. level to cool these stopes was an evident solution of the problem but a number of difficulties caused by the corrosive character of the water had to be overcome.

The air-conditioning plant constructed for the job is illustrated in Fig. 4. It was placed on the 3000 ft. level above the stoping area, and about 1000 ft. below the level on which the water entered the mine. The method of air conditioning employed is somewhat different from that used at the high-pressure plants, but it is one that has been in common use. Cold water entering the plant is first brought into contact with the air leaving the plant by being discharged from spray nozzles into the air stream. It then falls into a sump, from which it is pumped through another set of sprays, upstream as regards air flow, and this process is repeated until the water has passed through four sets of sprays in countercurrent heat exchange with the air. The discharged water is then turned into the mine drainage.

From the 1700-ft. level to the 2800-ft. level, the flow of the water is down through diamond-drill holes. From that point down to the 3000-ft. level, and in on that level to the plant, a distance of about 1200 ft., the water is carried in a line of 3-in. copper tubing. It was made up of a number of pieces of old copper heat-interchanger tubes, or second-hand material, which would have been sold as scrap metal. Joints were made by brazing near the end of each piece small rings cut from copper piping of standard size and were closed with a standard split-ring coupling.

The pumps used at the plant are of all bronze construction, and the glands are externally lubricated by fresh water from the mine supply system. All plant piping, spray nozzles, etc., are either copper or brass, but the baffle plates used to prevent spray water from being carried over by the air were made out of wood. It is not expected that the plant will indefinitely withstand corrosion of the water used, but present indications are that it will have a service life of several years.

Use of such corrosive water for air-conditioning purposes may be thought to be open to objection. However, since the temperature of the water is so much lower than that of the air, the cooling of the air causes the condensation of about 25 per cent of the water vapor originally carried by it, and there is no transfer of vapor from cold water to warm air. On Sept. 4, 1940, the volume of air passed through the plant was 32,000 cu. ft. per min. Initial temperature and wet-bulb were 89°F. and 85°F., respectively, and discharge temperature and wet-bulb were both 77°F. Volume of water passed through the plant was about 105 gal. per min.; initial temperature, 63°F.; discharge temperature, 82.5°F. Heat removed from mine air was approximately 17,500 B.t.u.

per min., or equal to the heat absorbed in the melting of 87 tons of ice per day.

The maximum improvement in any one working place was from a temperature of 89.5°F. (wet-bulb, 86°F.) to a temperature of 81°F. (wet-bulb 77°F.). The average improvement in conditions in seven stopes was from a temperature of 88.4°F. (84.9°F. wet-bulb) to a temperature of 84.0°F. and wet-bulb, 79.0°F. A number of other places were also benefited to a smaller extent.

At one of the main pump stations on a downcast shaft, much heat was transferred to air flowing into the mine by the electric motors driving the pumps. These older motors are cooled by air circulated through the windings. A small part of the water passing to the pumps was relatively cool so that the use of this water to absorb heat from the windings was naturally suggested.

At some places in the pump station, immediately over the motors, the conditions noted were: temperature 115°F., wet-bulb, 80.5°F. Since the sensible temperature of the air is very high, it is apparent that the quantity of water required to absorb the heat would be smallest if its temperature were increased to the highest possible degree by indirect countercurrent heat exchange with the air.

A duct system was installed, therefore, to collect air from the top of the pump station, and a fan and extended-surface type of heat absorber were installed, making an air-conditioning unit similar in general design to the 3900-ft. plant that has been described.

It was hoped to collect air from the motors at temperatures close to those mentioned, but the ducts, as now installed, have not accomplished this purpose, and some alterations and extensions will be necessary to prevent the infiltration of colder air from zones beyond the motors. However, conditions on the pump stations have been improved, and most of the heat formerly passed on to the mine has been diverted into the drainage water. Present operating results are as follows: Volume of air from pump station, 13,000 cu. ft. per min. Initial temperature and wet-bulb of air to heat absorber 87°F. and 79°F., respectively. Temperature and wet-bulb temperature of air discharged from the heat absorber, 66.5° and 66°F., respectively. Volume of water, about 75 gal. per min., initial temperature 59°F., discharge temperature 71°F.

As the mine was not operating at the time the plant was installed, no record of improvement in working places can be given. Conditions of the pump station were improved from temperature 115°F. (80.5°F. wet-bulb) to temperature 85°F. (wet-bulb 75°F.).

For the improvement of ventilation conditions in dead-end development headings, a number of small, portable air-conditioning units have been used, which may be operated with cold water supplied from the mine fresh-water system, or, occasionally, with cold water entering the drainage system. Two different designs have been developed, each of which has some advantages and some disadvantages.

The first portable units were made with an extended surface-heat absorber to transfer heat from air to water, and with a spray and baffle-plate dust scrubber to eliminate dust. The second type employs two or more stages of water sprays, or with direct contact between air and water, to effect the transfer of heat.

The heat-absorber type does not require any pumps, and water discharged from it may be piped to a higher level before being discharged to waste into the drainage system. It has the disadvantage of being heavy and easily damaged in being moved around the mine. Also, it is not easily cleaned.

The water-spray type of unit requires one or two pumps and the water leaving the plant must generally be turned into the ditch on the same level. It requires some

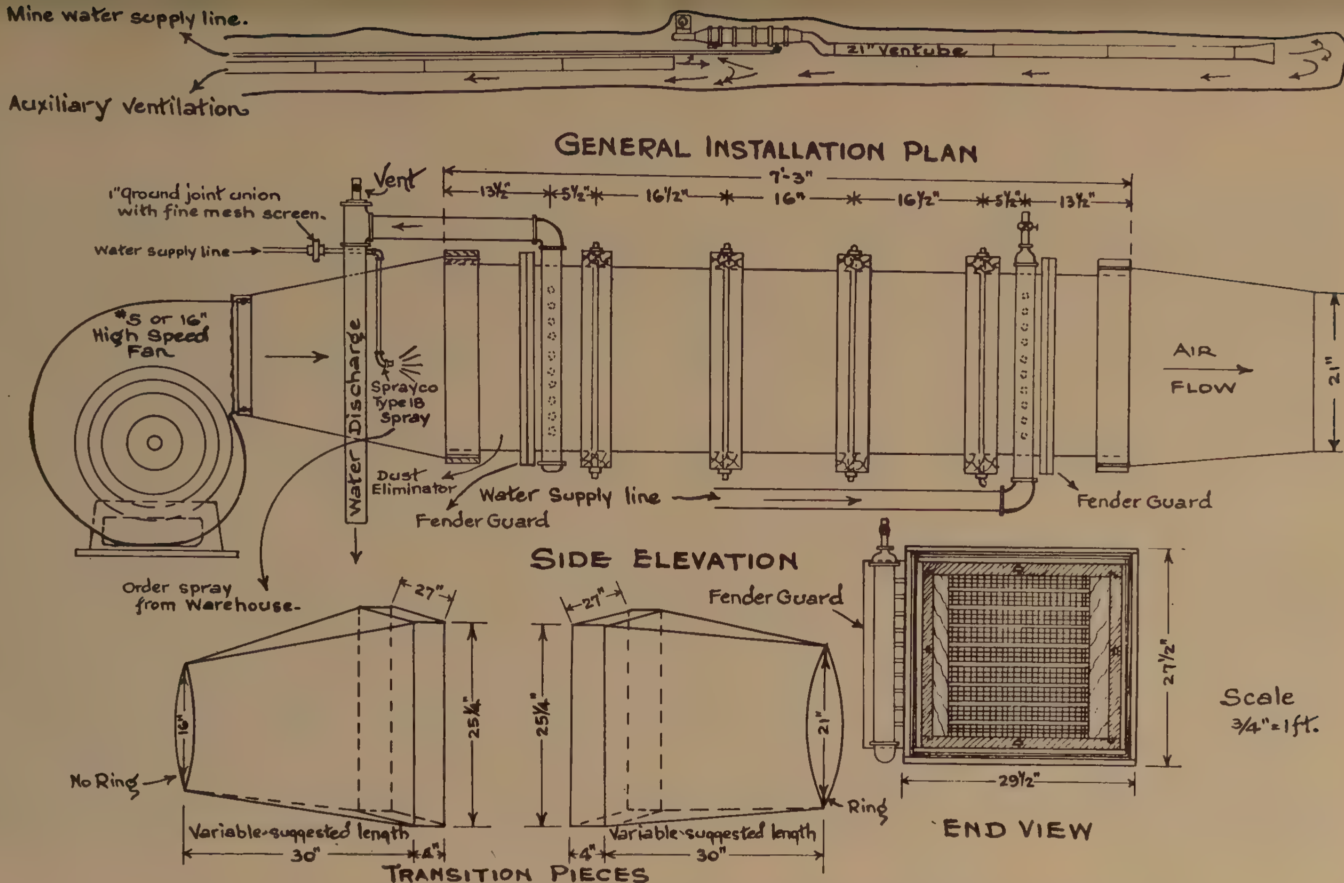

FIG. 5.—INSTALLATION SKETCH FOR FINNED HEAT ABSORBER.
Water flow must be countercurrent to air flow. Minimum water flow required, 15 gal. per minute. Heat transfer surface = 1042 sq. ft. per 10 coils. Fins = 0.0216 by 2 by 2-in. medium hard-rolled copper. Tubes = 5/8 outside diameter, 0.035-in. wall; annealed copper.

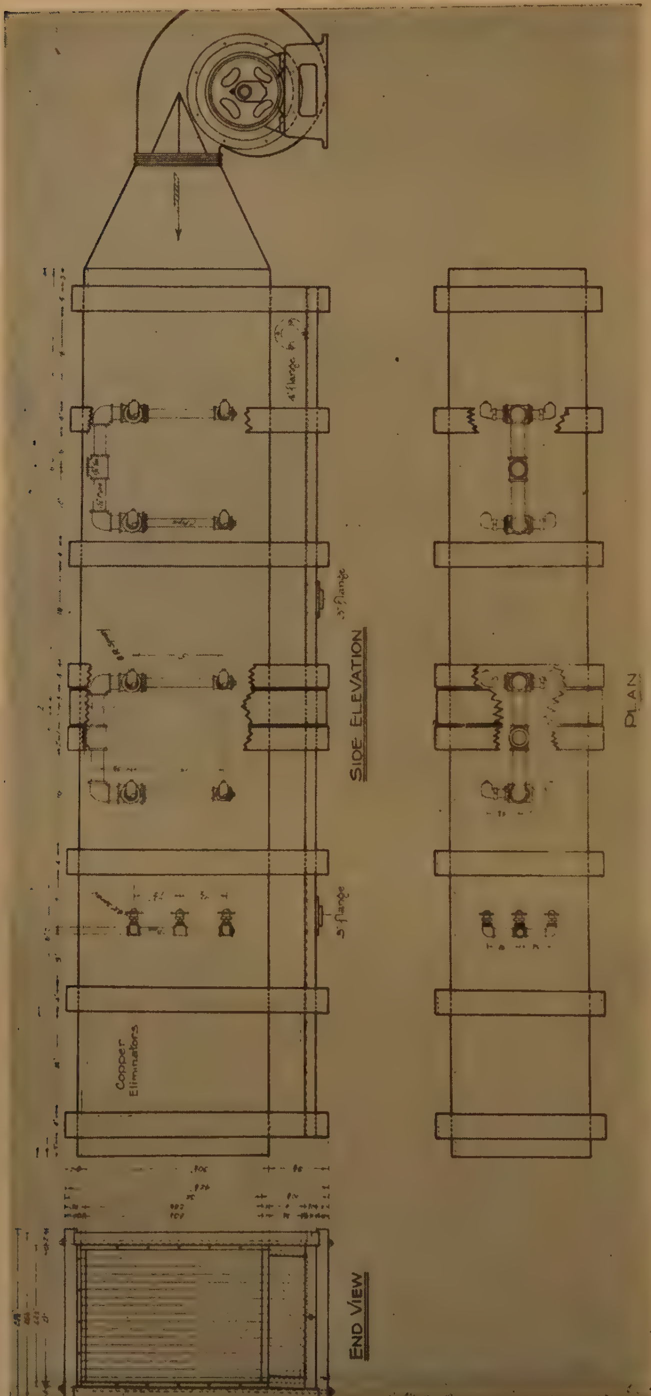

Fig. 6.—Three-stage portable air-conditioning plant.

attention to keep the pumps and sprays in good condition, but it is relatively light and is not so easily damaged in transportation.

A heat-absorber unit, with instructions for installation, is shown in Fig. 5. A unit of this type was used in cutting a pump station on the 3800-ft. level. Volume of air passed through the plant was about 4500 cu. ft. per min.; initial temperature and wet-bulb being 76°F., and discharge temperature and wet-bulb 61.5°F. The quantity of water put through the plant was about 20 gal. per min., the initial temperature in December being 45°F. and the discharge temperature under these conditions 62.5°F.

Before installation of this small plant, with auxiliary ventilation through flexible tubing from nearest fresh air source, the conditions at the working face were: temperature 90.5°F., wet-bulb 88°F. After installation of the plant, the conditions at the breast were: temperature 71°F., wet-bulb 70.5°F. The heat removed from the air was about 2800 B.t.u. per min., or equal to 14 tons of refrigerating effect.

Operating results with a two-stage, water-spray unit (similar to Fig. 6) were: Volume of air through the plant, 3000 cu. ft. per min.; initial temperature 91°F. (wet-bulb, 87°F.); discharge temperature and wet-bulb 75°F. Volume of water, about 39 gal. per minute; initial temperature 63°F., discharge temperature 71°F. Heat absorbed from the air about 2500 B.t.u. per min., or equal to 12½ tons of refrigerating effect.

The air from this plant was used to ventilate two raises. In one of them, the improvement was from temperature 90°F. (wet-bulb 87°F.) to temperature 82°F. (wet-bulb 81°F.). In the other raise the improvement was from temperature 90°F. (wet-bulb 87°F.) to temperature 83°F. (wet-bulb 81°F.).

Eleven low-pressure, portable units are in use. Of this number, seven are of the heat-absorber type and four are of the water-spray type. In addition, there are four larger water-spray plants, each of which has a capacity for cooling about 30,000 cu. ft. of air per minute, which are used for cooling small sections of mines where equipment for high-pressure, closed-circuit circulation plants is not yet available.

Installation of equipment for high-pressure, closed-circuit circulation plants at other mines is now under way. A surface cooling tower and 10-in. main-supply pipe lines are being installed at the Original mine. Main distribution pipe lines will be run on the 3800-ft. level to plants in the Steward, Anselmo, Original and Mountain Con mines. The plants to be installed will serve to cool and recondition about 400,000 cu. ft. of air per minute.

Comparisons of advantages and disadvantages of air-conditioning methods and ordinary mine-ventilation methods are almost impossible because both are necessary to a solution of the ventilation problem. The results obtainable by air-conditioning methods in the reduction of temperatures in some working places are not practicably obtainable by ordinary ventilation methods. On the other hand, some definite circulation of air through the mine, and equipment for the ventilation of dead ends, must be maintained even though air-conditioning methods are used. Air conditioning, or air cooling, is, therefore, essentially a supplementary measure for the improvement of temperature conditions on lower levels where ordinary ventilation methods are inadequate.

Economic advantages depend so much on rather variable conditions that it is difficult to make exact statements. Some advantages for air-conditioning methods are, however, evident. One of them is that air-conditioning equipment may be moved from one section of a mine to another, or, if necessary, from one mine to another, while many shafts, or other

main air courses, are of little value when certain sections of the mine are worked out, and expenditures for air shafts opened at one mine cannot be recovered and made available for the improvement of ventilation conditions at another mine.

Expenses for operating the air-conditioning plants are closely limited to power charges. A number of alterations and adjustments have been made to the existing plants, and repairs to mine timber at the plant stations have also been necessary; to some extent because these were the first plants built. However, for routine operation the only attention necessary is the cleaning of screens, which requires less than one hour per day and checks on operating performance, and no regular operators are employed.

Expense of ordinary ventilation includes a large number of items such as excavation of air courses and repairs, power to drive fans, flexible tubing for the ventilation of dead ends, and a number of miscellaneous charges. All of these costs must still be carried, at least to some extent, even though air conditioning be employed.

The heat removed from the working zone of a mine by ordinary ventilation may be fairly closely determined by the difference between the heat carried in the air entering the outlet airshafts and that carried by it as it enters the mine from the inlet shafts on the lower levels. On that basis, and under somewhat comparable conditions, the cost of power for the air-conditioning system, for equivalent cooling effect, was about 30 per cent of the cost of ordinary ventilation. Assuming that the expense of building the air-conditioning plants be absorbed in 10 years, the total cost of air conditioning, for equal cooling effect, would be about 75 per cent of that of ordinary ventilation. Some of the plants have now been in use for over five years, and should have an indefinitely long life.

The main benefit derived from the air-conditioning plants lies, of course, in the fact that a good average standard of working conditions has been maintained with mine production at full capacity, and this could not have been done without them.

Electric Blasting Practices of the Tennessee Copper Company

By R. G. Clay* and C. F. Seaman*

(New York Meeting, February 1942)

The mines of The Tennessee Copper Co. are in the Ducktown Basin, in southeastern Tennessee. The ore is a heavy sulphide consisting principally of chalcopyrite, pyrite and pyrrhotite and in places running as high as 95 per cent sulphur and iron. Operating mines are the Burra Burra and Eureka. Development of the Boyd ore body was started in 1940.

Late in 1939 the decision was made to change from fuse and cap to all electric blasting and at the same time electric cap lamps were installed to replace carbide. These changes were made to improve the safety and working conditions in the mines. Both the Bureau of Mines and explosives manufacturers recommended electric blasting as a safety measure.

Until that time, electric blasting had been used only for shaft sinking and in places where it was absolutely essential for safety. Very few men had had experience in this type of blasting, as the places where it was required were those that also required direct supervision. It was necessary for the mining staff to work out methods, procedure, and equipment to be used. In this they were ably assisted by representatives of the explosives manufacturers. Many mistakes were made and it is the purpose of this paper to describe the original plan and installation, show the major difficulties encountered, the remedies applied, and present the practices now being followed. Comparison of costs of the two methods will also be discussed.

About six months were required to make the change and eliminate fuse and cap blasting entirely. The first electric blasting stations were installed in November 1939 and the last fuse was taken underground in May 1940. A very unusual condition forced the use of fuse for a short time in one heading after May, but that will be discussed later.

Original Equipment and Installation

A power line is generally recommended as a source of current where permanent blasting stations are used. The plan worked out for electric blasting was to tap 220-volt alternating-current lines when they were available and use 250-volt direct current from the trolley wire at all other places. A blasting machine would be available at each shaft for places where there were no regular blasting stations and could be used if a regular station should fail.

The equipment decided on was No. 16 duplex reinforced lamp cord for lines to within about 50 ft. of the working place and No. 20 lead wire from that point to the shots. An open double-pole, double-throw switch with slate base was put in the circuit, so connected that the down position would close the line to the shots and the up position would connect it to the line from the power contactor. This switch was located in the circuit as close to the blast as was safe (should the shots explode when it was placed in the shooting

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position). In this way the man exposed to the explosion has the advantage of a shunted line while all connections are being made.

installed the equipment, completing one level before moving to another. The installation crew consisted of one engineer from the staff, one electrician, and one drill

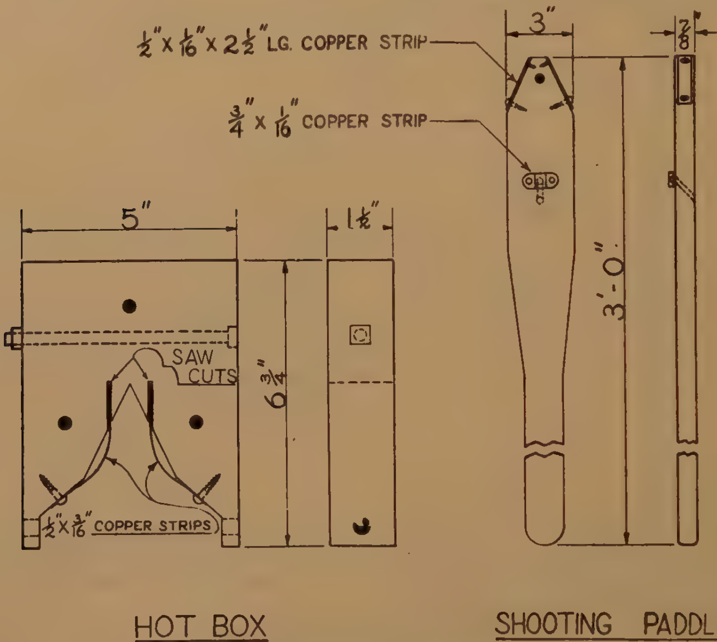

Fig. 1.—Details of contactor.

For power-line blasting, especially underground, an air gap of at least 18 in. is recognized as desirable. For alternating-current stations, the arrangement of paddle and hot box was used. Fig. 1 shows the details of the contactor. This arrangement was found satisfactory and is still being used for all power-line installations.

The direct-current contactor was arranged to contact the trolley wire. This did not allow for an air gap in the negative (return) side of the circuit and the ground wire was run directly from the rail to the shunting switch (referred to locally as the "safety switch"). This proved to be an unsafe practice and had to be changed, as will be explained later.

Because only a few of the crew were experienced in electric blasting, no attempt was made to rush the change. A small crew man. The engineer supervised the work, laid out each installation, and instructed the men in each place as the installations were completed. In this way, he was near by for several days following the change-over in any particular place. By the time all blasting stations on any given level were completed, the miners on that level were experienced enough to get along with such instructions as could be given by their foremen.

The original change-over required about 50 blasting stations for secondary breaking; 50 for stoping or primary breaking; and 25 for development places. Table 1 gives the cost of installing each of the two types of individual blasting stations. Holes are now drilled for the lines as development advances, at virtually no cost. During the original installation, however, these holes

were drilled as a part of the installation work.

TABLE 1.—*Cost of One Electric Blasting Station*

AVERAGE LENGTH 500 FEET FOR STOPE OR DEVELOPMENT WORK

28 holes at 50¢ each	$14.00
24 wood plugs at 5¢	1.20
1 shooting paddle	0.50
1 safety switch (open double-pole double-throw)	0.85
1 safety switch box	0.50
1 hot box	0.50
3 support plugs at 25¢	0.75
500 ft. reinforced lamp cord at 1.5¢	7.50
30 Nailit knobs at 0.5¢	0.15
Installation by electrician	6.00
Total cost	$31.95

AT A GRIZZLY OR SECONDARY PLACE

8 holes at 50¢	$ 4.00
5 wood plugs at 5¢	0.25
1 shooting paddle	0.50
1 safety switch (open-pole double-throw)	0.85
1 safety switch box	0.50
1 hot box	0.50
1 25-ohm resistor	0.85
3 support plugs at 25¢	0.75
100 ft. reinforced lamp cord at 1.5¢	1.50
10 Nailit knobs at 0.5¢	0.05
Installation by electrician	4.00
Total cost	$13.75

OPERATING COST

Figures are presented for two 12-month periods. The last half of 1938 and the first half of 1939 represent all fuse and cap blasting. The last half of 1940 and the first half of 1941 represent all electric blasting.

had to be blasted a second time because of missed holes.

To show the same cost for explosives with electric blasting that is attained with fuse and cap, there must be a saving in the amount of dynamite used. The over-all cost of electric exploders is about twice that of fuse and cap. Certain items of cost incident to electric blasting are more or less equalized by a like expense where fuse and caps were used. As an example, electric blasting requires the full time of one electrician. When fuse and cap were used, all exploders were made up on the surface and taken underground each day, which called for the full time of one man.

Other items of cost with electric blasting not classified as explosives are not offset by any similar cost when fuse and caps are used; for instance 5¼ tons of lead wire, and 4½ miles of reinforced lamp cord were used during the 12-month period covered by Table 2. These and other miscellaneous material items amount to a cost of $0.004 per ton for breaking and $0.095 per foot for development.

Adding this extra material cost to that shown in Table 2 for the period using electric blasting brings the breaking cost per ton to $0.076, or one mill per ton, total

TABLE 2.—*Cost of Dynamite and Detonators*

Period	Primary and Secondary Breaking			Development		
	Tons Broken	Cost of Explosives per Ton	Pounds Dynamite per Ton	Feet Developed	Cost of Explosives per Foot	Pounds Dynamite per Foot
1938–39	600,506	$0.075	0.51	14,726	$1.376	9.26
1940–41	752,979	0.072	0.47	13,878	1.690	11.22

There is a slight advantage in favor of the electric blasting for primary and secondary breaking, but this is accompanied by a decided increase in development cost per foot, which probably can be laid directly to general inexperience in electric blasting. A great many headings

cost, above the period for fuse and cap blasting. The development cost would be $1.785 per foot, or an increase of $0.409. Since all figures are based on the first 12-month period of electric blasting, it seems reasonable that in a second 12-month period the over-all cost of electric blasting

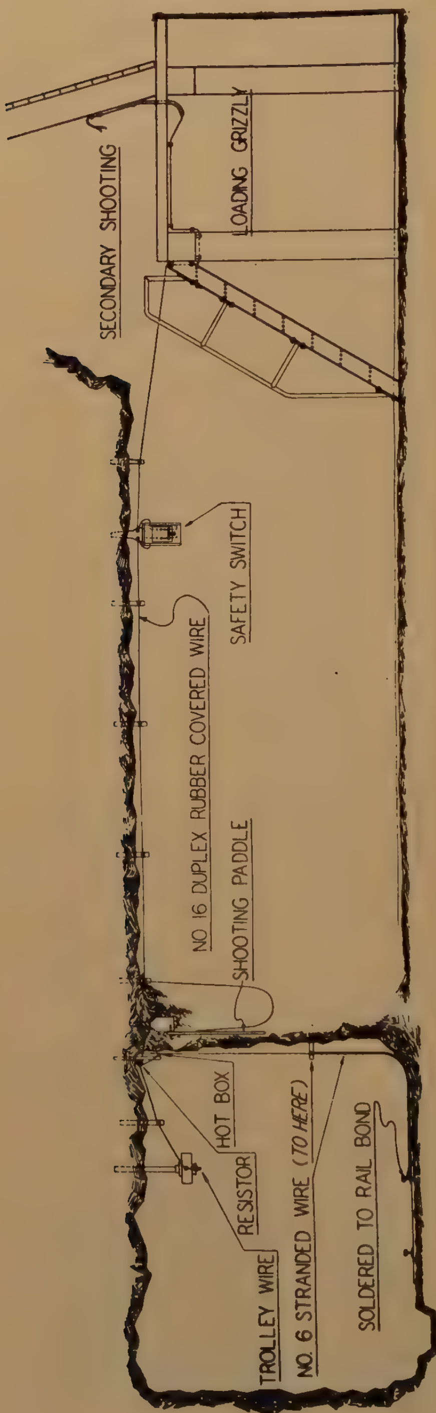

Fig. 2.—Secondary Blasting Station Using Direct Current from Trolley Wire.

may compare very favorably with fuse and cap blasting.

Troubles Encountered

The first trouble was caused by not having a break in the ground connection on the direct-current stations. All shooting circuits were comparatively short and there seemed to be no chance of a potential difference between the rail connection and the place where the shots would be fired. This assumption was erroneous, as was proved by a premature blast shortly after installations were started. In this particular place the ground connection to the rail was in a waste-rock drift and the shots were being fired in ore. One of the three shots exploded as soon as the safety switch was thrown into the shooting position.

In locating the trouble, voltage readings, taken with a voltmeter, showed a potential difference of from 6 to 8 volts between the rail (in waste) and the ore. What actually happened was that the blasting circuit had been accidentally grounded to the ore near the shots and the potential difference had caused enough current to flow to touch off the one shot. There are very few places in the mines in which this condition would be possible, but all direct-current stations were changed immediately to have the ground connection come through the contactor. This was accomplished by using the contactor designed for alternating-current stations and using the trolley wire for one side and the rail for the other. Fig. 2 shows a diagram of a secondary blasting station using direct current from the trolley wire.

Thirty-ampere fuses were installed on all stations originally to protect the line and contactors in case of a short circuit. These were found to be impractical on the direct current stations, as a ground on the circuit anywhere is equivalent to a short circuit. Air and water sprays, turned on before the blasts are made, would frequently short-circuit or ground the wires. The constant replacement and cost of refills

finally resulted in abandoning the use of fuses.

When fuses were thrown out, the condition that caused them to blow also caused the contactors on the direct-current stations to burn up. To eliminate this, a resistor was used to limit the flow of current on short circuit. The trolley circuit being 250 volts, a resistance of 25 ohms would limit the current flow to 10 amp. Experiments demonstrated that the contactors being used would permit the breaking of this current flow numerous times without serious damage. Using the resistor, it was found that the largest number of exploders desired for one shot could be blasted over the longest circuit in use. Calculations were as follows:

	Ohms
Permanent resistance ahead of plug......	25
30 electric blasting caps.................	37
1000 ft. No. 16 wire.....................	8
200 ft. No. 20 lead wire..................	2
Total resistance of circuit.............	72

From Ohm's law, electromotive force divided by resistance is equal to current flow: $250 \div 72 = 3.47$ amperes. The accepted minimum current for a series circuit is 1½ amperes, thus a factor of safety of better than 2 was maintained.

The type of ore being mined is very abrasive and in loading the holes extreme care must be used to avoid rubbing the insulation from the leg wires. It is also a very good conductor of electricity. This is demonstrated by the fact that in the heavy sulphide the circuit tester will not show any resistance when connected to the ore body.* This condition makes electric blasting more difficult than when the material being mined offers some resistance to the flow of an electric current.

In some places, it has been found advisable to check each round with a circuit tester, not only for continuous circuit but for connection to ground. If these checks show a ground to the ore, a process of elimination is used to determine the hole containing the grounded exploder. This exploder is cut out of the circuit, another is added and the test is repeated. No attempt is made to blast the round until it checks clear of ground. Enough tests were made in these places to determine definitely that successful blasting could not be done with sufficient ground to register on the circuit tester, even though a blasting machine or ungrounded source of current were used.

The conditions outlined forced discontinuance of the use of direct current from the trolley wire as a source of power for stope and development rounds. The negative or ground side of the circuit is at practically the same potential as the ore. This causes leakage and connections to the ore to be more pronounced than in an ungrounded circuit. A connection to ground ahead of the first exploder would cause the entire round to miss, and one farther along the circuit from the positive side would only put off the caps to that point. Leakage through the insulation of the leg wires in several holes weakens the current until a part of the round may fail to explode.

It is often difficult to determine definitely the cause of a misfire or blast failure, but indications are that in our mines 90 per cent are caused by grounds to the ore or leakage through the leg-wire insulation. No difficulty has been experienced from this cause when the blasts were in waste rock or nonconductive material. The other 10 per cent are from miscellaneous causes such as loose connections or a broken wire in the insulation of the main blasting line. Different types of leg-wire insulations have been tried and none of them have eliminatede lakage where used in heavy sulphide. If exploders are stored for any length of time underground and exposed to the moist mine atmosphere there is a marked deterioration in dielectric strength and resistance to abrasion.

* Mining Methods of the Tennessee Copper Company. U. S. Bur. Mines *Inf. Circ.* 6149.

In driving the 2500-ft. long waste-rock crosscut from the Burra to the Boyd ore body, an unusual condition forced a return to the use of fuse and cap blasting for a short time in that heading. The face of the crosscut was near the Boyd ore body and small stringers of mineralization were being cut, when the drillers complained of feeling shock while drilling. Voltmeter readings, taken between a drill rod in a hole and the drill jumbo, showed a fluctuating potential difference up to 26 volts, and indicated a flow of direct current from the rock to the rail. The track into the heading was not bonded, a battery locomotive having been used for haulage. The only electric line in the crosscut was the 220-volt alternating-current blasting line with the contactor located in the Burra ore body.

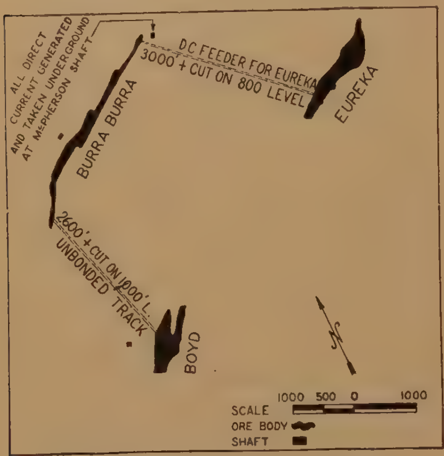

Fig. 3.—Relation of ore bodies.

Tests indicated that the high fluctuating potential was from the return circuit of the Eureka mine haulage system. Direct current for the haulage motors in all the mines is furnished by one generating unit on surface near the McPherson shaft. This shaft serves the Burra ore body and all direct current is taken underground at that place. The haulage motors at the Eureka mine are served from a feeder through the 3000-ft. long crosscut from the north end of Burra to Eureka on the 800-ft. level. The Boyd crosscut on the 1000-ft. level is 3000 ft. along the Burra strike, south of the one to Eureka. The Eureka and Boyd ore bodies are about 4000 ft. apart and are on the same general strike, but there was no known connection between them. The return circuit for the Eureka haulage system is a 40-lb. bonded track through the Burra to Eureka crosscut. The relation of these ore bodies is shown in Fig. 3.

In addition to the fluctuating potential difference, there was 1.5 volts constant difference when the generating unit was shut down. Geophysical work done before the Boyd crosscut was started indicated that the Boyd and Eureka ore bodies were at the same potential and that they were approximately 1.5 volts above the Burra ore body. No thought was given to this condition while the crosscut was being driven, but it would have been dangerous even without potential difference from the haulage circuit.

The attempt to ground the rails to the rock failed to eliminate any of the potential difference and the crosscut was extended into the ore body by the use of fuse and cap blasting. When well into the Boyd ore body, the rails were bonded through the crosscut and grounded to the ore body. Electric blasting was resumed and no further trouble has been experienced from that source.

The possibility of such an occurrence at any place where a crosscut is driven between two separate conductive ore bodies should be stressed. While there was no premature blast at the place described, conditions were right for a disastrous accident.

Present Practices

Electric exploders are hauled underground once each week and those left over from the preceding week are brought back to the surface. For this transportation

two boxes are used (Fig. 4) for each level. Each box holds a supply for one week. The exploders left in the boxes brought out are sorted, put in special bins to dry, and then reissued. They remain at least a week in the surface magazine before being sent back underground. During this time a large part of the moisture absorbed by the leg-wire insulation while underground has dried out.

The series connection has been adopted as standard and is used exclusively. It was desirable, especially when starting electric blasting, to use one standard method of connections, although in a few instances some other connection might be more efficient. The series connection has four advantages: (1) A blasting machine can be used, (2) a large number of exploders can be fired from a great distance with a small wire, (3) connections are simple to make and to explain, (4) the circuit is easily checked with a circuit tester. Series wiring has the very great disadvantage of small cumulative leakage, which causes many misfires. As already mentioned this single disadvantage of the series is largely responsible for 90 per cent of all misfires, but this is more than offset by the advantages stated above.

The source of power almost always used for secondary breaking is the trolley wire (Fig. 2). The use of a resistor ahead of the contact box makes this method very satisfactory.

Fig. 4.—Box for transporting electric exploders underground.

A few stopes are still using the trolley-wire current but the general practice is to use 220-volt alternating current. Where this is not available, a blasting machine is used. In some of the stopes, especially those in very heavy sulphide, each round is checked with a circuit tester. The retreating method of mining allows the miner to shorten his main shooting wire in the stope as it becomes bad near the shots.

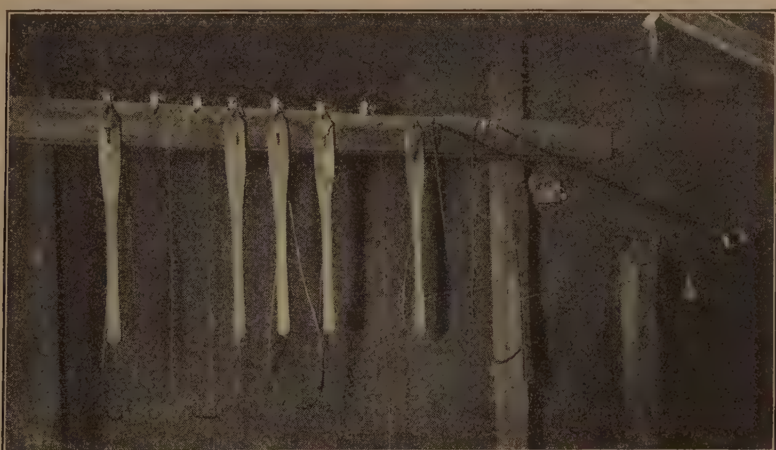

Fig. 5.—Station for stope blasting where several circuits use same hot box.

The safety switch usually is in the subdrift, near the manway, and the shooting contactor or blasting machine is on the main level. Fig. 5 shows a station for stope blasting where several circuits use the same 220-volt alternating-current hot box for power. All paddles are plainly marked for the places they control.

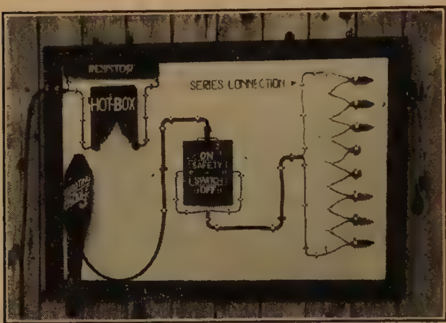

Fig. 6.—Demonstration board.

Blasting machines are used in nearly all development work. Alternating current was found satisfactory for this work, but usually is not available. The same standard and arrangements are used for the blasting machines as for a power circuit. The only difference is that the wire ends are prepared for attaching to the blasting machine instead of being fastened to the shooting paddle.

It is difficult to maintain a good shooting line in a raise. The present practice is to use lead wire in raises up to 150 ft. Occasionally some of it can be used more than once. In raises of more than 150 ft., shooting lines are extended into the raise. They are put in behind the pipe lines and, as far as possible, protected from the falling muck.

Since each miner does his own shooting, the demonstration board shown in Fig. 6 has been found useful. It is used in meetings with the foreman and crew to explain the circuit and demonstrate how connections are made, and how the contactor should be used.

General Instructions Issued as Standard for Electric Blasting

1. Electrician will install all blasting stations.

2. Standard stations will consist of a box with spring contacts, contact paddle, safety switch and No. 16 rubber-covered line. On direct-current installations a 25-ohm resistor will be placed on the positive side ahead of contacts. The safety switch will be type 3061 open double-pole, double-throw, installed with the shunted position down. Rubber-covered line will be hung on insulators and must not come in contact with the rock, pipe lines or hangers.

3. All shots made from power circuits must be from standard shooting stations.

4. Shots in temporary places will be made with a blasting machine. Both wires must be disconnected from machine and twisted together at all times except when firing round. Permanent places using blasting machine shall have permanent line and safety switch.

5. No one circuit shall have more than 30 exploders, without special arrangements.

6. The safety switch shall be kept in the down, or shunt position at all times except when putting off shot.

7. When extensions are required (the point where more than 150 ft. of lead wire is needed) drillmen will make them with rubber-covered wire. It will be hung on insulators and not allowed to touch rock, pipe lines or hangers.

8. Electrician will repair shooting equipment once each month. He will clean and repair all contacts, tighten and repair connections, follow line to the end of rubber-covered wire and make any necessary repair, and finally check for a complete circuit.

9. Type 3061 switches shall not be used underground for any other purpose than shunting or safety switches.

10. Foremen shall have electrician remove shooting equipment when station is

no longer needed. As long as it is left in place it must be repaired each month.

Conclusions

In summarizing the findings of one year of all electric blasting, a few direct statements are in order. These are all based on experience at the mines at Ducktown and may not apply under other conditions.

1. A slight increase in over-all cost may be expected with electric blasting.

2. A large amount of training is required for the crew.

3. For general work in which many men connect shots, the series connection is best.

4. For power-line blasting, a switch is not safe for underground use. The insulation between the blasting circuit and the source of power should be an air gap of at least 18 inches.

5. The firing contacts should be so arranged that power cannot be left on the shooting circuit accidentally.

6. The blasting circuit should be as short as is practical.

7. Blasting lines should be kept short-circuited at all times except when shots are actually being fired. This may be done with a switch located as near the shots as is safe.

8. The blasting lines are an electric circuit and all connections should be carefully and correctly made.

9. The blasting machine is a very satisfactory source of power.

10. Ungrounded alternating current is the best source of power.

11. For secondary breaking and such places as require only a small number of shots, direct current from the trolley wire can be used.

12. Where direct current from the trolley circuit is used, it is essential that the rail side of the circuit be considered as charged, and contact made only at firing time.

13. By far the greater number of misfires are caused by leakage of the leg-wire insulation.

14. No trouble is caused by leg-wire leakage when blasts are in waste rock.

15. The insulation on the leg wire will absorb moisture and lose some of its dielectric strength and resistance to abrasion if stored underground for any length of time.

16. Fuses cannot be used on a blasting circuit.

17. In driving a crosscut between two separate conductive ore bodies, extreme care must be used and a constant check made for stray currents when nearing the mineralized zone.

18. Working conditions are better with electric blasting than with fuse and cap because of: (1) elimination of fuse smoke; (2) elimination of waiting period in case of blast failure; (3) elimination of need for man to hold light for shot firer in secondary breaking.

19. It ensures electric blasting being used wherever safety demands it.

20. A constant check is necessary with electric blasting, just as with fuse and cap blasting, to see that all safety standards are maintained and carried out.

During the 10-year period immediately preceding the adoption of electric blasting at Ducktown, there were two blasting injuries attributed to use of fuse, and several near accidents. No injuries have been received since electric blasting was begun, and no near accidents, other than those discussed, have been reported.

DISCUSSION

(Jay A. Carpenter presiding)

B. F. Tillson,* Upper Montclair, N. J.—I want to emphasize that the paper shows that there are hazards connected with electric blasting, although it replaced fuse blasting as a safety measure recommended by U. S. Bureau of Mines and by explosives manufacturers.

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* Consulting Engineer.

In my association with fuse blasting, I recollect only one accident from it in 25 years. In a western mine where detonators and a coil of fuse were separately issued to miners instead of being crimped on at a central fuse house, my buddy exploded a detonator in careless handling—possibly a spark from a cigarette—and lost several fingers from one hand.

Of course my men have had accidents from drilling into explosives left in old drill holes (some in old workings mined years before), but I cannot classify such accidents as definitely due to fuse blasting.

Although also using electric blasting, and having a part in the early use of delay-action electric exploders and fuse igniters, I have found many causes for failure and potential hazards associated with them, for example:

1. Blasting machines easily get out of shape in the rough and tumble of mining and from the humidity of mine air, and rarely exhibit their rated capacity.

2. There is not a close enough standardization of the bridge wire resistances and heating to prevent some electric exploder from detonating and not the others in series with it, and if in parallel this condition is aggravated.

3. The maintenance of sufficient insulation of lead wires to exploders to prevent misfires requires a high order of supervision, and the abrasion of insulation in loading holes is another risk to circumvent.

4. The hazard of premature explosions from stray currents (either from voltaic effect of ore body, poor grounding of electric haulage, or power feeders, or induced currents, from lightning strokes, which may be carried into the mine).

5. The damaged copper wires in the muck pile may not only prove a nuisance but become a hazard if later tossed over the conductors of a high-tension transmission line (as happened when one of my men was electrocuted).

6. Simultaneous blast of a number of holes may produce unsafe roof rock.

In contrast, what is the case against the use of fuse?

1. The smoke is objectionable where ventilation is insufficient.

2. Smouldering fuse may be cast into nearby timbering and start a mine fire.

3. Fuse may be cut off by preceding hole and cause a misfire with detonator and explosive in the muck pile, or old drill hole.

4. Lack of skill of blaster may get him blasted by cutting fuse the wrong lengths, or failing to ignite them in proper order and expeditiously.

C. F. Jackson,* Washington, D. C.—Several years ago, I had occasion to visit the underground workings of the Spruce mine at Eveleth, Minn. I noticed that in blasting down top slices they were employing cordeau and inquired why they did not use electric blasting. The reason given was that a survey had shown the presence of stray electric currents within the ore body running up to several amperes in strength and that the presence of these stray currents would make electric blasting very hazardous.

The underground workings are very close to electric-shovel operations in the Spruce pit on one side and the city of Eveleth on the other, where much electrical equipment, household appliances and the like are in use.

While I have long been an advocate of electric blasting where a large number of shots are to be fired in one round, especially in shafts, raises and confined stopes, the situation at Eveleth indicates that electric blasting is not always advisable.

* Chief, Mining Division, U. S. Bureau of Mines.

Blasting Practices at the New Cornelia Open-pit Copper Mine

By Harry H. Angst,* Member A.I.M.E., and Reuel A. Cochrane†

(New York Meeting, February 1941)

The successful exploitation by opencut methods of the low-grade porphyry copper deposits is due to the economical handling of large tonnages. Large tonnages are possible only if the rock material is broken so that the large power shovels can operate with minimum delays. This is particularly true at the New Cornelia mine, where the dense, hard, tough rock material creates problems not encountered in most of the porphyry copper mines. The introduction of churn drills at the New Cornelia for primary drilling, in 1936, necessitated a complete change in the methods of breaking ground as formerly air drills had been used.

The blasting operations are of particular interest because of the handling of great quantities of high explosives in safely and efficiently fragmenting the hard rock material.

Rock Formations

At the New Cornelia mine the rocks are quartz monzonite with a diorite border facies intruding a rhyolitic series. Overlying these formations is a fanglomerate formation composed of boulders of monzonite, diorite, rhyolite, and other types of rock of unknown derivation. (The word "fanglomerate" is used in this paper in referring to the local conglomerate type of rock formations.)

The ore body is approximately 4800 ft. long and 2800 ft. wide, and is of the disseminated type. Mineralization occurs principally in the quartz monzonite and to some extent in the adjacent diorite and rhyolite. The principal ore minerals are chalcopyrite, bornite, and a small amount of chalcocite.

Over a considerable area the quartz monzonite is extremely hard and siliceous in character, owing to pegmatitic zones of alteration. The hardness of the ground depends on the intensity of the silicification. In a small area in the pit, the monzonite contains an abundance of pyrite accompanied by sericitization and is soft and easily fragmented. This is true also in the remaining oxidized portion of the ore body.

The diorite and rhyolite formations, like the monzonite, vary from extremely hard to very soft material, but on the whole are easier to fragment than the monzonite.

Jointing is prevalent in all the rock formations and plays an important role in the blasting efficiencies. In the fanglomerate formation, numerous well-defined joints, widely spaced, tend to make the ground blocky when blasted. This is a deterrent to good fragmentation. In the other rocks the joints are more numerous and are closely spaced, except in localized areas where the formations are extremely hard.

These descriptions indicate that the lowest blasting efficiency will be obtained in the harder formations, from which the major portion of the mine production is obtained; namely, the monzonite and rhyolite.

All rock formations mined at the New Cornelia pit are so hard that excavation

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TABLE 1.—*Explosives Used at New Cornelia*[a]

Description	Strength, Per Cent	Velocity, Ft. per Sec.	Cartridge, Size and Weight	Standard Uses at New Cornelia Mine
L.F. SPECIAL GELATIN EXPLOSIVES				
Special gelatin stick powder. This explosive contains ammonium nitrate ingredients and is a gelatin type. It is very plastic and has a high velocity that transmits a good shattering effect if it is closely confined within the material to be blasted. Water resistance in waxed-paper containers is 72 hours.	40	12,467	1¼ by 12 in. ¾ lb.	1. Used in blasting all primary *air-drill* blast holes *containing water*, in formations ranging from leached quartz monzonite to medium-hard quartz monzonite. 2. This is the only type of explosive used in the secondary blasting operations at the New Cornelia mine.
	60	15,420	1¼ by 12 in. ¾ lb.	1. Used in blasting all primary *air-drill* holes, *wet* or *dry*, in formations ranging from hard quartz monzonite through fanglomerates, dense rhyolites, to extra-hard quartz monzonites. 2. Used in all springing operations.
Special gelatin bulk powder. The ingredients of this explosive are the same as the Special Gelatin stick powders but it is packaged in large containers equipped with special hangers, in order that it may be handled by a powder-loading machine developed at the New Cornelia mine.	40	12,467	8 by 18 in. 50 lb.	Loaded by powder-loading machine into *churn-drill* blast holes. This explosive is used in formation ranging in hardness from medium-hard quartz monzonite or fanglomerate to hard silicified quartz monzonites.
	60	15,420	8 by 18 in.	Loaded by powder-loading machine into *churn-drill* blast holes. This explosive is used in all churn-drill blast holes, wet or dry, in formations ranging from hard quartz monzonite, dense rhyolite and fanglomerate to the extra-hard quartz monzonite.
L.F. AMOGEL EXPLOSIVES				
No. 6 Amogel powder. An ammonium nitrate explosive of semi-gelatin, slight cohesive nature	30	Approximately 8,000	8 by 27 in. 50 lb.	1. Loaded by powder-loading machine into *churn-drill* holes *containing water*, in soft formations such as altered quartz monzonite.
No. 4 Amogel powder. The Amogels have a medium-high velocity and contain 33 per cent more bulk per corresponding weight when being compared with the Special Gelatin. Water resistance in waxed paper containers is 24 hours.	40	11,811	8 by 27 in. 50 lb.	1. Loaded by powder-loading machine into *churn-drill* blast holes *containing water*, in altered quartz monzonite formations of medium hardness. 2. Used as a deck load in *churn-drill* blast holes, *wet* or *dry*, in formations ranging from medium-hard to hard silicified quartz monzonite.
L.F. QUARRY EXPLOSIVES				
Quarry No. 4. Quarry explosives are free flowing and for this reason are especially efficient when loading air-drill blast holes. The Quarry explosives are composed of granules approximately the size of sand, which flow freely through blast holes that are as small as ⅞ in. in diameter. This explosive is similar to Amogel in that it contains 33 per cent more bulk per corresponding weight when it is compared with the Special Gelatin explosives.	30	5,085	No. 16 bags. 12½ lb.	1. Used in both *air-drill* and *churn-drill* blast holes in *dry*, soft formations of leached monzonite.

TABLE 1.—(*Continued*)

Description	Strength, Per Cent	Velocity, Ft. per Sec.	Cartridge, Size and Weight	Standard Uses at New Cornelia Mine
Quarry No. 1. The use of Quarry explosives is definitely limited in the Cornelia blasting operations because the water resistance of the explosive is nil. Definite savings are effected in dry formations where this type of explosive can be used.	60	6,234	No. 16 bags. 12½ lb.	1. Used in both *air-drill* and *churn-drill* blast holes in *dry*, soft to medium-hard, formations of leached monzonite.
Ajo Special Explosive. Ajo Special was developed especially for use in the New Cornelia mine. It is semi-free flowing and is approximately the same consistency as whole wheat flour. Ajo Special is not water resistant, therefore its use is governed by the same restrictions that control the use of Quarry explosives.	70	9,500	No. 16 bags. 12½ lb.	1. Used in blasting of both *air-drill* and *churn-drill* holes in *dry*, medium-hard formations of quartz monzonite.

[a] Information listed in this table is given only as it affects the New Cornelia pit-blasting operations and does not necessarily conform with the recommendations of the explosive manufacturer.

without blasting is impossible. The softest formation encountered in the blasting operation is the leached monzonite capping overlying the sulphide ore, but this is small in amount when compared with the total tonnages blasted.

The natural difficulties encountered in safely and economically fragmenting these silicified ore and waste-rock formations have largely been solved by the following procedure:

1. Detailed study and experimentation in the development of modern drilling and blasting procedure.
2. The invention and development of efficient equipment to aid in the safe handling of large quantities of explosives.
3. The formulation of detailed plans for each operation.
4. The development of a carefully trained drilling and blasting organization.

Explosives

The rock formations of various types and physical character naturally require different types and amounts of explosive to fragment them properly. For this purpose, three entirely different types of explosives, involving nine variations, are used in the blasting operation. All dynamites used are manufactured by the Apache Powder Co. at Benson, Ariz., and pertinent information regarding their description, strength, and uses are given in Table 1.

Storage of Explosives

Transportation of explosives from the manufacturing plant to Ajo is made by freight car via the Southern Pacific and the Tucson, Cornelia and Gila Bend Railroads. Upon arrival at the Ajo depot, the railway cars containing explosives are immediately routed to either of the two explosive-storage magazines at the New Cornelia mine. The combined capacity of the two magazines is twelve thousand 50-lb. boxes, or 600,000 lb. of explosives.

Both of the storage magazines have a safety factor exceeding Government and State regulations pertaining to the location and construction of such buildings. Each magazine is completely "bullet proof," containing a steel door and adobe brick walls 18 in. thick, with 1½ in. of cement plaster on each side of the walls. Strongly braced ceiling board 3 in. thick supports 8 in. of ground-rock insulation that is topped by a roof constructed of galvanized iron. The magazines have efficient ventilation and a daily log of temperatures proves

that the temperatures are very uniform. Explosives are stacked in such a manner as to promote ventilation and to facilitate efficient removal, which is governed by the age of the explosive (Fig. 1).

FIG. 1.—INTERIOR OF POWDER MAGAZINE, SHOWING ROLLER CONVEYOR.

DRILLING AND SPRINGING OPERATIONS

The New Cornelia mine contains 8.6 miles of shovel benches. Prior to the year 1936, all primary drilling was with tripod and wagon air drills on benches about 30 ft. high. When the wagon drills were replaced by churn drills, these benches were consolidated into benches varying from 50 to 60 ft. in height. In the year 1938 it was proved that from the standpoint of the drilling and blasting, shovel loading, and railroad-track operations a bench height of 40 ft. was most efficient. Since that time, deepening of the mine pit has been with 40-ft. benches. The reasons leading to adoption of this practice are as follows:

1. Use of blast-hole churn drills has proved to be most economical and efficient on benches not exceeding 40 ft. in height.

2. Benches 40 ft. high afford the most satisfactory shovel tonnages in facing up bench cuts. The power shovels in use at the New Cornelia pit have a boom height of 36 ft., therefore facing up of benches 40 ft. high can be done with greater efficiency and less danger than is encountered in facing up benches that are higher than 40 feet.

3. The power shovels in use at the New Cornelia pit take a cut averaging 65 ft. in width. After the cleanup cut has been taken from a bench 40 ft. high, the bench can be blasted with the railroad track in position. The blast lays the broken material up to the track in such a manner that the power shovel can return and make the splatter cut without removal of the track.

At the present time, 90 per cent of the ground is drilled by churn drills and the remaining 10 per cent by wagon drills. The wagon drills are used in drilling low benches resulting from approach grades to the various pit levels.

All primary drilling operations are conducted on benches that have been leveled off by a bulldozer and "faced up" by shovel operations. All primary blast holes are drilled to a predetermined depth below shovel grade. The distance drilled below the shovel grade varies from $\frac{1}{10}$ to $\frac{1}{6}$ of the total depth of the blast hole, depending upon the distance from the blast hole to the toe of the bench.

STAKING DRILL HOLES AND PLACING EXPLOSIVE LOADS

The importance of correct location of holes to be drilled in preparation for the blasting operation cannot be overemphasized. Correct location has a definite bearing on blasting costs and its importance is only exceeded by that of selecting the correct type of explosive and computing the amount needed to fragment the rock formation efficiently.

Each hole to be drilled is accurately located and the spot is indicated by a metal marker, inscribed with all necessary drilling and sampling information. After being placed, these hole markers are not disturbed until the drill machine is set up

directly over them, ready to commence drilling. The average bench-face slope is 70°.

toe, and air-drill toeholes are drilled to fragment the extra toe burden. These holes are drilled into the bench toe on a

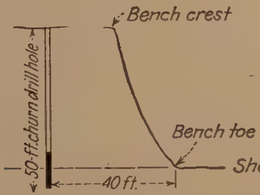

Fig. 2.

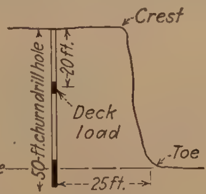

Fig. 3.

Fig. 2.—Benches having face slope ranging between 60° and 70°.
Fig. 3.—Benches having face slope ranging from 70° to vertical.

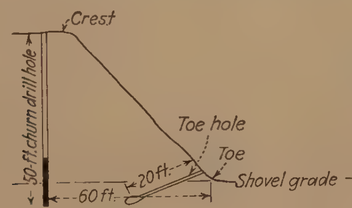

Fig. 4.

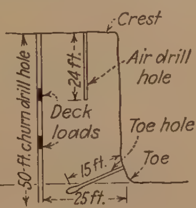

Fig. 5.

Fig. 4.—Benches having face slope of less than 60°.
Fig. 5.—Fanglomerate formations having an almost vertical face.

Table 2 gives the normal distances used as a basis in staking benches of average slope in rock formations of varying hardness.

Table 2.—*Normal Distances Used as Basis in Staking Benches*

Formation Hardness	Height of Bench, Ft.	Depth Drilled below Grade, Ft.	Distance between Holes, Ft.	Distance from Toe of Bench, Ft.	Number Times Each Hole Sprung
Air-drill Holes					
Soft	20	4	12	12–16	1
Medium	20	4	11	12–16	1
Hard	20	4	9	12–16	2
Extra hard	20	4	8	12–16	3
Churn-drill Holes					
Soft	40	6	14	23–35	0
Medium	40	6	14	23–35	0
Hard	40	6	12	23–35	0
Extra hard	40	6	10	23–35	0
Soft	60	8	18	35–50	0
Medium	60	8	16	35–50	0
Hard	60	8	14	35–50	0
Extra hard	60	8	12	35–50	0

Benches having a face slope of less than 60° are regarded as having an abnormal horizontal slope ranging from 5 to 20° and their length is regulated to fragment the bench toe extending beyond the normal bench-face slope of 70°.

Benches of hard rock having a face slope of more than 70° and a height ranging from 40 to 60 ft. are ideal for the use of deck explosive loads to fragment the upper portion of the bench. Deck explosive loads are small explosive loads placed in the barrel of the blast hole to aid in fragmenting and dropping the material contained in the crest of a high bench.

Placement of drill holes and explosive charges in benches of different face slopes is illustrated in Figs. 2 to 5, and placement of lines of bank holes to straighten an irregular bench is shown in Fig. 6.

Wagon Drills

All primary blast holes not exceeding a depth of 24 ft. are drilled by air drills with wagon mounting. The drill machine proper weighs 165 lb. and contains a piston of 4-in. diameter. This drill machine is

mounted on the conventional type of three-wheel wagon-drill carriage. The drill steel consists of 1¼-in. round, threaded, hollow drill steel in 4-ft. changes. This drill steel

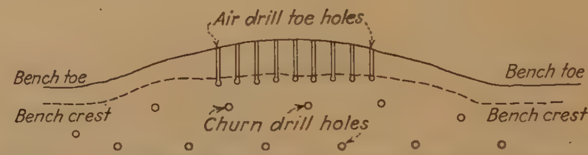

FIG. 6.—PLACEMENT OF TWO LINES OF VERTICAL BANK HOLES IN ADDITION TO TOEHOLES TO STRAIGHTEN AN IRREGULAR BENCH.

is equipped with detachable bits ranging in size from 3 in. down to 1½ in. in diameter. The detachable bits are resharpened in the mine bit shop, an average of seven sharpenings being obtained from each bit.

Springing.—Before being sprung, the average 24-ft. air-drill hole has a diameter of 2½ in. at the collar and tapers to a bottom diameter of 1⅞ in. The average 24-ft. air-drill hole is staked to fragment 65 cu. yd. of material. Because of the hardness

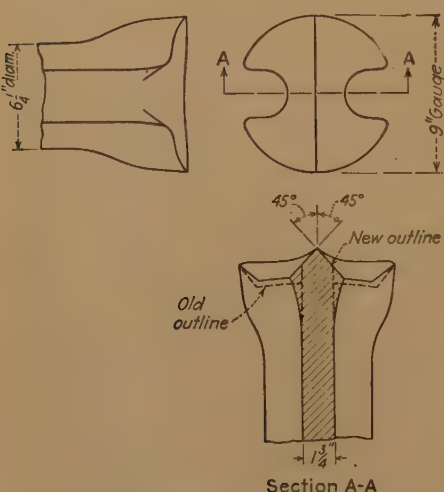

FIG. 7.—CHURN-DRILL BIT, AJO TYPE.

of the New Cornelia rock, all air-drill holes exceeding 12 ft. in depth are sprung. The number of springs per hole varies from one to three, depending upon the hardness of the rock formation and the depth of the hole. When the hole is to be loaded, the 4 ft. of blast hole below shovel grade must contain at least two thirds of the total load. In order to create the required cavity below shovel grade, springing operations are carried out as follows, using 60 per cent Special Gelatin 1¼ by 12-in. stick powder:

Spring No.	Below Shovel Grade, Lb.	Above Grade	Stemming, Flotation Tailings, Ft.
1	5	0	4
2	12	0	8
3	30	0	16

No. 8 electric caps are used for detonation of all springing charges and the charges are composed of 60 per cent Special Gelatin stick powder.

The excessive heat generated by the explosion of a springing load is retained for a long period of time in all dry rock formations in the New Cornelia pit. To prevent any possibility of a premature explosion caused by this heat, the following rules are strictly enforced:

1. A blast hole must not be sprung twice in the same day.
2. A blast hole must not be sprung and loaded the same day.

Churn Drills

All blast holes deeper than 24 ft. are drilled by the latest type of churn drills, which are electrically powered, mounted on caterpillar treads and handle a string of drilling tools weighing 2700 pounds.

The cutting face of the churn-drill bit has a distinctive design developed at the mine bit shop (Fig. 7). The cutting face

has a diameter of 9 in. and is particularly efficient in drilling hard rock formations.

The average churn-drill hole is 9¼ in. in diameter from collar to bottom and this large diameter renders springing unnecessary because the hole readily accommodates sufficient explosive below shovel grade to successfully fragment the toe of an average bench.

Ten-inch casing pipe is used to case through any broken material encountered in starting a hole. The remainder of the churn-drill hole is not cased. Churn-drill casing is not removed before blasting, as it is seldom damaged by the blast.

Loading Practices and Accident Prevention

Transportation of Explosives from Storage Magazines to Mine Pit

The average primary blast at the New Cornelia mine consumes 20,000 lb. of explosives and breaks 100,000 tons of rock, although blasts requiring 50,000 lb. of explosives are not unusual. The largest bank blast fired to date at the mine contained 102,050 lb. of explosive and broke 382,832 tons of rock.

All portions of the mine are connected by a well-maintained network of supply autotruck roads. All portions of the mine pit may also be reached by railroad track car or train.

When a blast of large size is to be loaded, efficient and safe delivery of large quantities of explosives is made by use of the railway powder car (Fig. 8).

Delivery of explosives to loading zones for average size shots is generally by powder supply truck (Fig. 9), by powder supply track car (Fig. 10), or by use of both the autotruck and the track car.

Small amounts of explosives, to be used in secondary blasting of boulders, are stored in portable explosives magazines stationed at convenient points within the mine pit. Daily delivery of explosives to the portable magazines is made with supply track car and trailer (Fig. 11). All explosives not used are returned to the magazines at the end of the day shift.

Computing Explosive Loads

The explosive load is computed for the overburden on each blast hole. The overburden is the rock material contained between the blast hole and the bench face. This is the only material figured in computing the explosive load. However, the material broken behind the line of churn-drill holes substantially increases the tons broken per pound of powder used.

The rock material broken behind the line of blast holes is referred to as the "backbreak" and generally ranges from one-half to all the tonnage broken in front of the blast hole. The type and weight of the explosive load per cubic yard of overburden is determined by the hardness and structure of the material to be blasted. Table 3 explains the loading factors used.

TABLE 3.—*Loading Factors*

Formation Hardness	Formation Condition	Explosive Used	Weight of Explosive per Cubic Yard of Burden, Lb.
Very soft......	Dry	No. 4 Quarry	½
Soft...........	Dry	No. 1 Quarry	⅝
Medium.......	Dry	No. 1 Quarry	¾
Medium hard..	Dry	Ajo Special	⅞
Hard..........	Dry	40 % Special Gelatin	1
Extra hard....	Dry	60 % Special Gelatin	1–1¼
Very soft......	Wet	No. 6 Amogel	½
Soft...........	Wet	No. 4 Amogel	⅝
Medium.......	Wet	No. 4 Amogel	¾
Medium hard..	Wet	No. 4 Amogel and 40 % Special Gelatin	⅞
Hard..........	Wet	40 % Special Gelatin	1
Extra hard....	Wet	60 % Special Gelatin	1–1¼

Primary Blasts

Large primary blasts are loaded and fired only during daylight hours. Loaded ground is never left unguarded and is fired as soon

as loaded, irrespective of the time of day. Stringent rules cover movement of men and materials in and adjacent to the loading zone and such movement is closely

4. Supplies of explosives are delivered and spaced in such a manner within the loading zone as to preclude any possibility of a chain explosion (Fig. 12).

Fig. 8.—Railway powder car.
Fig. 10.—Powder supply track car.
Fig. 9.—Powder supply truck.
Fig. 11.—Delivering explosives to portable magazines by supply track car.

supervised by the powder boss in direct charge of the loading operation. The following procedure is followed in the loading and firing of all primary blasts.

1. The explosive load per blast hole is calculated as to type and amount needed.

2. An area surrounding the blast holes is cleaned, marked, and restricted as a loading zone. The loading zone is completely enclosed with a fence constructed of rope of a bright red color. This restricted zone is also indicated by red flags large enough to be clearly seen from any portion of the mine pit.

3. Powder-loading equipment is delivered to the loading zone.

5. Powder crews are spaced so as to carry on their duties at a distance of at least 30 ft. from the next nearest powder crew.

6. The primacord is attached to the first explosive cartridge lowered into the blast hole. After the cartridge is firmly seated on the bottom of the hole, the primacord is cut at a point 2 ft. above the collar of the hole and tied to a piece of weighted wood, to prevent the upper end of the primacord from being pulled into the blast hole. The remainder of the bottom charge is then lowered into the hole.

7. The hole is stemmed to the point of the deck load (if one is needed). The deck load is threaded on the primacord

and lowered to rest upon the stemming material. The remainder of the hole is then filled with stemming. The primacord is constantly checked to prevent the upper

Fig. 12.—Loading churn-drill blast holes with powder-loading machine.

end from being pulled into the hole by the settling of the stemming material.

8. When all holes are loaded, a mainline primacord is strung from end to end of the loading zone and the primacord protruding from each blast hole is tied securely to this mainline primacord.

9. Watchmen are sent to levels above and below the loading zone to clear men and materials from the blast zone when signaled.

10. To signal the impending blast, a blasting signal is blown by a very large air whistle at the train dispatcher's shack, on a small hill overlooking the entire mine pit. This whistle is audible to every employee in the pit.

11. A blasting signal is blown from a whistle at the loading zone to signal the exact location of the blast.

12. The watchmen on the levels surrounding the blast clear all men and materials from the blasting zone and when flag signals have been received from these watchmen that the entire blast zone is clear, one man at each end of the loading zone connects an electric detonating cap to his respective end of the mainline

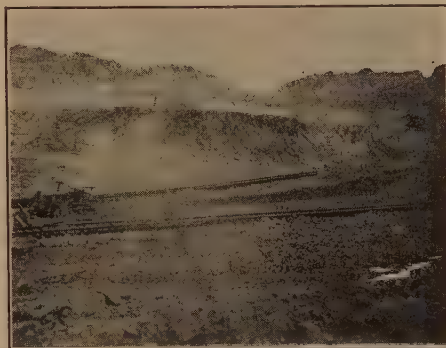

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Fig. 13.—Bench with blast holes loaded and ready for blasting.
Fig. 14.—Same bench during blast.
Fig. 15.—Same bench after completed blast.

primacord. When these men, one of whom is the powder foreman, have retired from the blast zone, double check signals are given, and if all is clear the powder foreman signals the firing of the blast.

The electric caps that detonate the blast are fired by battery current.

All electric wires to the blast are laid and final connections are made by a competent electrician especially trained for this kind of work.

13. After the blast has been fired and the dust and gases have cleared, the blast is very closely checked for the possibilities of a missed hole.

Figs. 13, 14 and 15 show a bench before, during and after blasting.

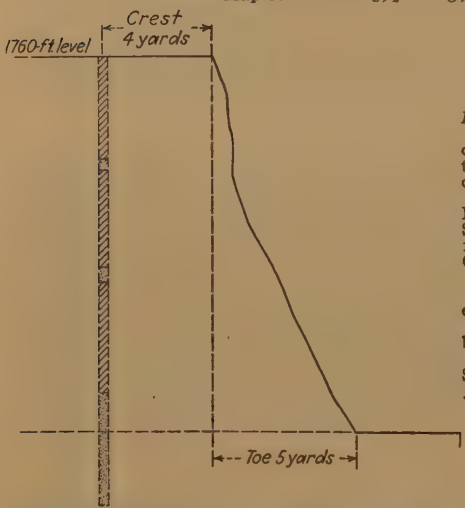
Complete records of all primary blasts are kept. Samples of these records are shown in Tables 4 to 6.

Secondary Blasting

All boulders that will not pass through the shovel dipper are set back of the shovel and left to be drilled with jackhammers

TABLE 4.—*Record of Primary Blast, Bank Profile View*

Illustrated record of explosive load in blast No. 178. Bank No. 1700–1760. Map coordinates O½-6 to O½-3½. Date blasted March 23.



Procedure followed in loading this blast.

Note—Sketch in all extra holes drilled in bank such as crest holes, toeholes, etc. Write all information pertaining to the loading of holes on this page or the following page of this report.

Bank Hole Type, Churn drill. Depth in Yards, 22. Bottom Charge, 7½ Boxes. Type, 40 % Special Gelatin Stemming above Bottom Charge, 6 lineal yards. 1st Deck Charge 1½ Boxes. Type, 40 Per Cent Special Gelatin Stemming above Column Charge, 4 lin. yards.

Remarks

A second deck charge was placed at the 18-ft. depth in each churn-drill hole.

This deck charge was to fragment the material in the bench above the material affected by the first deck charge.

2nd Deck Charge, 1 Box. Type, 40 Per Cent Special Gelatin. Stemming above 2nd Deck Charge, 5½ lineal yards.

Formation Type, Quartz Monzonite. Hardness, Hard.
Average Explosive Load, Entire Blast 3/10 Lb. per Cu. Yd. Overburden.
Tons Loaded Average Shovel Shift, 3900. Opinion of Blast, Very Good.
Signed R. A. Cochrane.
Title: Foreman Drilling and Blasting Department.

No missed primary blast holes have been encountered in two years at the New Cornelia pit, but should one occur it is the direct duty of the powder foreman to supervise: (1) cleaning of the missed hole; (2) resumption of excavation activities in the vicinity of a missed hole that has been cleaned.

Plans for clearing men and materials from the danger zone surrounding a blast are made in advance. These movements are so well coordinated that the delay to mining operations in the blast area rarely exceeds 10 minutes.

(Fig. 16). The blasting of these boulders comprises the largest portion of secondary blasting activities at the New Cornelia pit.

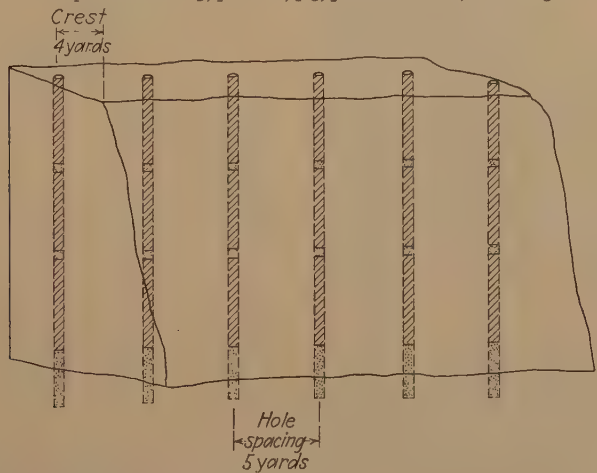
More powdermen are required in secondary blasting than are used in primary blasting. During the year 1939, out of the total of 40 powdermen used in all blasting operations, 22 were engaged in secondary blasting. These men loaded and blasted 119,600 lb. of explosive, which broke 215,000 boulders—an average of 700 boulders for each day.

Boulders are loaded with 40 per cent Special Gelatin stick powder which is detonated with No. 8 electric caps if the boulders are in clusters, and by No. 8 fire-fuse caps if the boulders are scattered.

When boulders are to be blasted with electric caps, these caps are wired up in series, each series not to exceed 10 caps, and are fired by the blasting electrician, who uses an alternating electric current circuit of 220 volts for ignition of the blast.

TABLE 5.—*Record of Primary Blast, Bank-face View*

Illustrated record of explosive load in blast No. 178. Bank No. 1700–1760.
Map coordinates Q½-6 to O½-3½. Date blasted, March 23.



Appearance of Bank When Blasted: Fragmented very good.
Engineer's Estimate of Tonnage Broken, 150,000 tons.
Actual Tonnage Dug from This Blast, 147,000 tons.
Tons per Pound of Explosive Used, 5.

Remarks

Difficulties Encountered in Drilling, Blasting, or Shovel Loading While in This Blast Area. This was the fourth consecutive test blast in this area. Much difficulty had been experienced in satisfactorily fragmenting the material and creating a definite line of backbreak due to numerous formation slips.

The following is a comparison between the first and the fourth test blasts:

First Test Blast	Fourth Test Blast
3.8 Tons per pound powder	5. Tons per pound powder
Poor fragmentation	Fragmentation very good.

Signed R. A. Cochrane.
Title: Foreman Drilling and Blasting Department.

The safeguarding of secondary blast zones is similar to that used in primary blasting, except that only one warning whistle signal is used and that whistle is placed at the location of the boulders to be blasted. It indicates the location and the number of boulders to be blasted.

All blasting work is closely supervised and no powderman is allowed to light over eight fuses on rough ground, ten fuses on smooth footing, or to light even one fuse unless accompanied by another powderman.

All electrical secondary blasting is done in daylight hours, but fire-fuse blasting is necessary throughout the 24 hr. of each day the mine is in operation.

All secondary blasting is so closely coordinated with the mine operation that very few operating delays are caused by the performance of this work.

Stemming and Detonation of Blast Holes

Until July 1930 a leaching plant for the treatment of oxide ores was in operation at Ajo. The residue of this process, leach tailings, is being utilized as blast-hole

stemming in the present-day blasting operation. The tailings are loaded by power shovel from the tailings dump into railway dump cars, then hauled to the mine pit and dumped into a gravity feed bin. blast hole. Tamping of explosive or stemming in the blast holes is not necessary. Each blast hole is stemmed to the collar of the hole.

Primacord, a detonating fuse that has a

TABLE 6.—*Primary Blast Record, Blast No.* 178, *March* 23, 1940

Coordinate location (Q½-6 to Q½-⅞)		
Level, ft	1,700	
Hole number	(1 EE-12 EE)(1 M-36 M)	
Type of material	Monzonite	
Number of churn-drill holes	51	
Average height of bench, ft	59	
Average depth of holes, ft	67	
Average depth below floor, ft	8	
Maximum depth of holes, ft	67	
Total drill footage	3,417	
Feet drilled per shift	48.8	
Total drill shifts	70	
Number of bits used	168	
Feet drilled per bit	20.3	
Hole spacing	5	
Distance from crest	4	
Distance from toe	5	
Tons broken	150,000	
Pounds powder used	29,300	
Tons broken per pound of powder	5.1	
Tons broken per foot drilled	43.9	
Loading calculation, lb. per cu. yd	7/10	
Powder per hole: springing	None	
Bottom charge	22,450	
Deck load	6,850	
Spacing of load in hole, ft. from top of hole:		
1st deck load	30	
2nd deck load	18	
Primacord	4,500	
Costs		
1. Drilling:		
70 drill shifts @ $31.04 per shift	$2,172.80	
168 drill bits @ $ 1.53 per bit	257.04	
Casing uses @ $ per use (est. at 0.0012 per ton broken)	180.00	
Transportation, eight truck shifts	118.72	
Repairs (est. at 0.00268 per ton broken)	402.00	
Supervision, eight shifts	118.72	
Total drilling		$3,229.28
2. Blasting:		
1,650 lb. powder per blast @ 0.09694¢ (33 boxes Ajo Special)	$ 159.95	
27,650 lb. powder per blast @ 0.095¢ (553 boxes 8 × 18 40 %)	2,626.75	
4,500 ft. primacord @ 0.03797¢	170.87	
2 electric caps @ 0.0993¢	0.20	
Stemming (tailings haulage, 2½ shifts)	25.93	
Supervision and loading	108.53	
Total blasting		$3,092.23
Total costs		$6,321.51
Cost per foot of drilling	$ 0.9509	
Cost of drilling per ton broken	0.0217	
Cost of powder per ton broken	0.0186	
Cost of Cordeau per ton broken	0.0011	
Other blasting costs per ton broken	0.0009	
Total cost per ton broken	0.0423	

Dump-truck haulage is used to convey the stemming material from the storage bin to the blast holes (Fig. 17). The leach tailings are 3-mesh and are greatly decomposed and softened in character.

When shoveled into the blast holes the stemming settles under its own weight to create a seal, which has shown no tendency to release any portion of the explosive gases through the barrel of the velocity of 20,350 ft. per second, is used exclusively in the detonation of all primary blasts. This velocity exceeds by nearly 5000 ft. the explosive velocity of any type of powder used in the blasting operation.

As the primacord is in contact with all explosive contained in the blast, its velocity propagates the explosion throughout the entire blast virtually instantaneously. This has a definite effect in the achievement of

better fragmentation, owing to increased explosive efficiencies created by the high velocity of the initial detonation.

Powder-loading Machine

At the New Cornelia mine, the policies of the Phelps Dodge Corporation in regard to accident prevention take form in a close-knit program consisting of liberal safety education of the employee combined with constant improvement of operating machinery and operating methods. An enviable safety record has been established by following a code of safe practices that exceed the ordinary rulings pertaining to blasting operations.

One outstanding mechanical improvement in the safe handling of explosives has been the invention and development of a powder-loading machine (Fig. 18). This machine is "foolproof," having "Deadman" control, and can be safely operated by men untrained in its use. A safety limit of over 7 to 1 was carried out in the construction of the machine. Primarily it was intended as a safety measure to eliminate the dropping of high explosives into deep churn-drill holes. When this safety measure was accomplished, it was discovered that the following operating benefits were also obtained by use of the mechanical loader:

1. A reduction of 30 per cent in the time consumed by manual loading.

2. A better seal of the explosive loads due to the use of a 50-lb. explosive cartridge, which eliminates the voids that previously occurred with the smaller cartridge used in manual loading.

3. Ten per cent more explosive in the vital portion of the blast hole below shovel grade because the large cartridge eliminates voids.

4. Disappearance of powder headaches among the employees because the explosive container, which released harmful fumes into the open air, no longer needs to be opened.

The mechanical loader can be described briefly as a light, portable crane equipped

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17

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Fig. 16.—Boulders set aside to be drilled with jackhammers.
Fig. 17.—Haulage truck dumping stemming material at blast hole.
Fig. 18.—Powder-loading machine.

with a short boom, powered by a small air motor and mounted on a framework supported by two pneumatic-tired wheels. The powder-lowering medium is a suitable length of ⅜-in. dia., high-test air hose

connected to a specially designed hook with air release.

This loader successfully handles explosive cartridges weighing 50 pounds.

In the manual loading operations before introduction of the machine, explosive cartridges weighing 12½ lb. were cut into three pieces to be dropped into the churn-drill blast holes.

Occurrence of Water in Drilling Operations

Impervious rock formations retard drainage of benches in the mine, and water in considerable amounts is encountered in 80 per cent of all blast holes drilled. Because of the hard, siliceous rock formations, approximately half the small number of blast holes that do not contain water cannot be blasted with free-flowing Quarry explosives because of the added bulk and lessened velocity of the latter as compared to the Special Gelatin explosives that are used in this type of ground. When dry rock formations are encountered that permit the use of the Quarry explosives, considerable savings are effected by such use. If this water problem did not exist in the New Cornelia pit, added savings could be effected by the use of low-cost, free-flowing Quarry explosives in the softer formations.

Record of Blasting Efficiencies at New Cornelia Mine for the Year 1939

During the year 1939 the following explosives supplies were consumed at the New Cornelia mine:

1,000,000 lineal feet, or 189 miles, of primacord
200,000 lineal feet, or 38 miles, of fire fuse
59,000 No. 8 fire-fuse blasting caps
30,000 No. 8 electric blasting caps
2,554,900 lb., or 1277 tons, of bulk explosives

Over 14,000,000 tons of hard and siliceous waste and ore-rock formations were blasted in consuming the explosives listed. For each pound of explosive used, 5.48 tons, or 10,960 lb., of ore or waste rock were broken.

Operating conditions ranged from mountainside waste removal to the blasting of a circular drop cut 40 ft. deep and 3000 ft. long.

The mine operated 309 days and 180 men, as an average, were employed in the drilling and blasting operations.

For each primary powderman man-shift, 460 lb. of explosive were loaded, stemmed and blasted, and 2520 tons of rock were properly fragmented.

A reserve of blasted ore and waste rock equal to the amount necessary for one month of operation is kept ahead of the power shovels.

The tonnage of ore and waste rock drilled and blasted per month equals that dug by the power shovels. The average amount of ore and waste rock mined per month during the year 1939 was 1,250,000 tons.

The average month of operation at the New Cornelia mine during the year 1939 resulted in the following:

30,000 ft. of primary chrun-drill holes blasted
8,500 ft. of primary air-drill holes blasted
7,500 lineal feet of mine bench broken
228,000 lb. of explosive consumed

Blasting of churn-drill holes resulted in 40.04 tons being broken per foot of drill hole compared with 5.96 tons broken per foot of air-drill hole blasted. Each churn-drill hole averaged 2002 tons of broken rock.

The storing, transporting, loading and blasting of this large quantity of explosives was efficiently accomplished without a serious accident of any kind, which is a tribute to the careful planning and supervision of this operation and a credit to the employees of the blasting department. The powdermen are all Mexicans and

Papago Indians, who have been well trained in their duties and are "safety conscious" at all times.

At present, 37 per cent of the total mine organization is engaged in breaking-ground operations and 43 per cent of the total mine operating cost per ton is that of breaking grcund. Although the blasting operations are quite satisfactory, constant efforts are being made to improve the efficiency in breaking ground and thus make it possible to mine more economically the hard rock encountered in the New Cornelia pit.

Detachable Rock-drill Bits at the Hollinger Mine

By Aloys H. Wohlrab,* Member A.I.M.E.

(New York Meeting, February 1942)

The conditions that govern the selection of a suitable type of detachable bit for the small, isolated mine, for rock work and tunnel contracting and for the large mine are quite dissimilar, therefore intelligent recommendation of a particular make of detachable bit must involve consideration of many factors. Small, isolated mines may find advantageous a detachable bit that a larger mine cannot afford to use, because of more efficient conventional steel sharpening and steel-distribution practice, but a small mine may not be able to adopt the practice of large mines because of necessary capital expenditures for equipment for detachable-bit sharpening and shanking. The perfect detachable bit, suitable for all conditions, has not yet been developed.

Conventional Drill Bit in Use

That drilling speed increases as the bit gauge decreases is well known, but since small-hole drilling is the exception rather than the rule the rate of such increase is probably not so generally recognized. Tests carried out by Holman Brothers Limited[1] with 7/8-in. steel show that when bit sizes are reduced from 1 1/2 in., 1 3/8 in., and 1 3/16 in., to 1 3/8 in., 1 1/4 in., and 1 1/8 in., respectively, the volume of the hole is 15 per cent less and the drilling speed 49 per cent higher. Recent tests by the United States Bureau of Mines[2,3] in hard, uniform basalt showed that a reduction in average gauge from 2.077 to 1.826 in., a difference of 0.251 in., gave an increased drilling speed of 65 per cent.

The limits to which a drill hole may be extended depend upon the loss in gauge. Since the conventional bit with 5° and 14° clearances did not possess sufficient reaming edge to permit the smaller gauge changes that were required for small-hole drilling, experiments were made in bit construction with a view to correcting this deficiency. This resulted in the adoption of the full reaming cross bit (having full Carr characteristics) by some mines.[4]

In order to drill holes having a minimum taper from the collar to the bottom of the hole, full reaming center-hole cross bits with 90° cutting angles, 5° and 14° wing tapers, forged on 7/8-in. quarter-octagon drill rods, are used in the conventional drill-steel practice at Hollinger. In the steel shop close attention is given to the condition of the dies and dollies, which, combined with a rigid inspection of the finished bit, ensures as nearly a perfect bit as can be forged economically. Shop practices and costs have been discussed in earlier papers.[5,6]

The method by which the radii for the special dies used for forging bits of different gauge diameter are obtained so that the points and reaming edges cut circles of the same diameter has been discussed by Hibbert.[7]

Requirements for Detachable Bit for Hollinger

Under these circumstances, it was apparent that it would be difficult to introduce a detachable bit that could successfully compete with the conventional bit already in

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[1] References are at the end of the paper.

use. However, if a detachable bit of equivalent performance could be developed that would cost no more than the conventional bit, an annual saving of $50,000 in direct nipping and transportation costs would be secured. Some savings of an intangible nature, and others that cannot be computed, will be mentioned later.

Low first cost of the manufactured bit and shank was desired. Since the weakest link in all types of detachable bits is the connection of the bit to the rod, it was considered necessary to retain the full reaming characteristics of the conventional bit already in use, thus keeping the excavation per foot of hole drilled at a minimum and lessening the strain at this point. Experiments in hot milling, grinding, forging and turning in a lathe followed by broaching showed that the latter method not only automatically provided more accurate gauge sizes but that with quantity production bits could be reconditioned at a lower cost. Most of the commercial detachable bit connections, when reduced in size to permit the use of bits of as small a gauge as was required and subjected to repeated blows delivered by a 3½-in.-diameter drifter machine at 90 lb. air pressure, showed up inherent weakness due to the method of attaching the bit to the rod. Hence it was necessary to devise a different method of attachment.

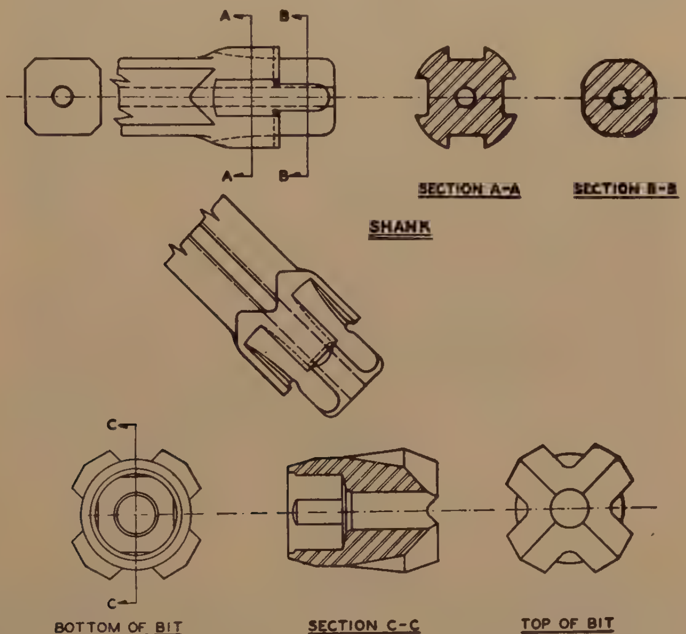

FIG. 1.—LIDDICOAT TYPE B DETACHABLE BIT AND BIT CONNECTION.

Another requirement of the attachment was that the cost of manufacture be low in order that the over-all cost per bit use should approximate that of the conventional forged bit.

LIDDICOAT TYPE B DETACHABLE BIT

The Liddicoat type B detachable bit (Fig. 1), which is used at Hollinger, is a full reaming, center-hole, cross bit made of straight carbon steel having a 0.90 to 0.95 per cent carbon content. The center hole is of ample diameter to permit unobstructed flow of water to the bottom of the drill hole. Only one heat and two operations are re-

quired to completely fashion the bit. The heated slug is extruded under pressure into a die and the flashings are trimmed off in a punch press. The cutting edges of the bit are retouched by grinding, the bit is hardened and is ready for market. All new bits are 1⅝-in. gauge and when worn are reduced 1/16 in. for successive gauge changes in the reconditioning plant, thus allowing for a total of five changes to 1⅜ inches.

FIG. 2.—BIT EXTRACTOR AND BIT CARRIER.
Each carrier will hold 60 bits.

The socket at the base of the bit is a driving fit on the end of the rod or shank. This end of the rod is rounded but has four milled sides, which drive between the slightly flattened projections in the otherwise round socket, thus preventing the bit from turning on the rod. As the bit is driven farther back on the rod by the blows of the machine when drilling, the lower taper on the outside of the bit skirt, which has been left in an annealed condition, is driven back behind four undercut lips on the rod, thus crimping the skirt of the bit to the rod. If from repeated use the male end of the rod becomes worn and the bit is loose on the rod, the crimped skirt gives added insurance that the bit can be withdrawn from the hole. The bit is readily removed from the rod by placing it in the bit extractor (Fig. 2) and striking the wedge piece *A* a few smart blows with a 5-lb. hammer. A very cheap and efficient substitute for this type of extractor is a piece of 5-in. shafting weighing about 10 lb., with a hole drilled through the center too small to clear the wings of the bit. When placed over the shank end of the rod and dropped against the wings of the bit this weight will remove it easily. Because the latter type of extractor might accidentally come up with the ore and cause a broken crusher shaft, it is not used at Hollinger.

Reconditioning Plant

The dull bits as well as the sharp ones that have not been used during the shift are brought up from underground in holders (Fig. 2) and are taken to the sorting and racking room. The dull bits are replaced by sharp ones of the same diameter from stock and the holder is ready to be taken back into the mine. The dull bits are inspected for defects and breakage, counted and tabulated on the day's run, after which they are dumped 400 to 500 at a time into a W. W. Sly Manufacturing Company's rotary shot-blast machine, where all dirt and rust are removed and the bits are made clean and bright. This process is essential, since any quartz or grit left on the bit dulls the cutting tools on the broaching machine and on the lathe. The

clean bits are next placed in a cylindrical pot, which holds about 1300 bits, and are

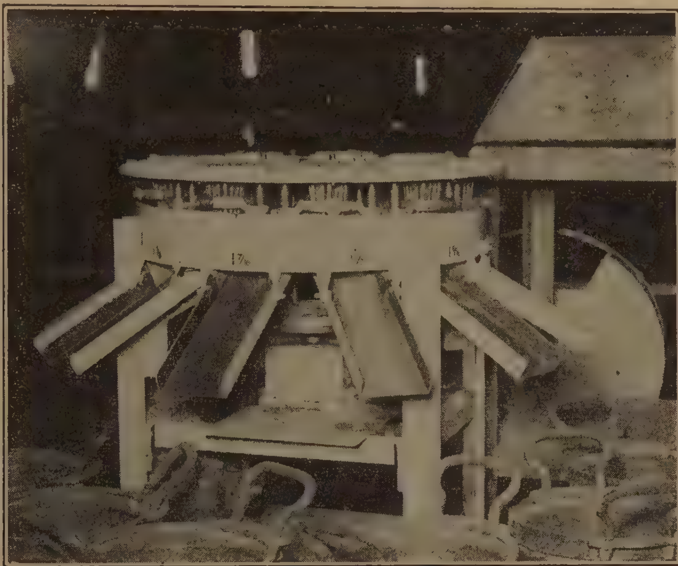

FIG. 3.—ROTARY BIT SELECTOR OR CLASSIFIER.

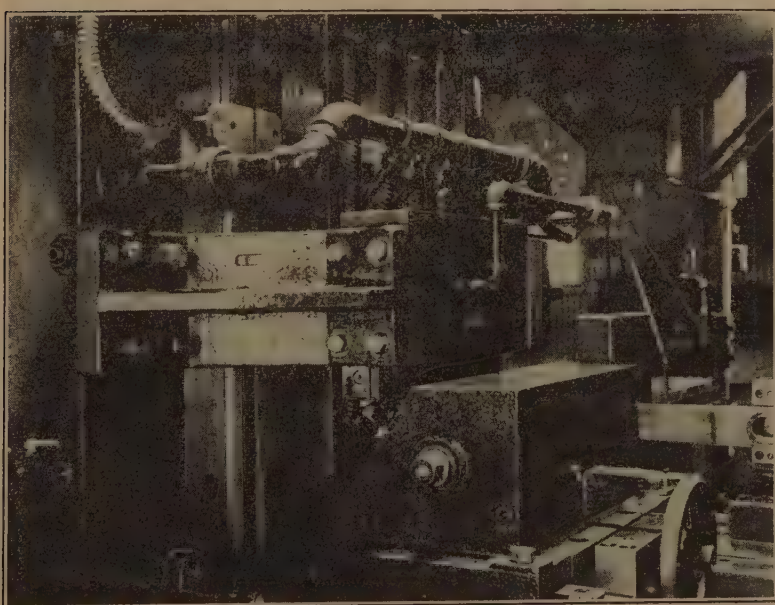

FIG. 4.—SLIDING TABLE AND BROACHES.

lowered into a thermostatically controlled Lindberg Engineering Company's electric annealing furnace, where they are allowed to soak at an annealing temperature and then to cool gradually. Following annealing, a motor-driven rotary selector automatically classifies the bits according to gauge and places them in separate pots (Fig. 3).

Gauging is performed on a Leblond 11-in. rapid production lathe equipped with a follower operating on a cam from a spring loaded slide, which gives the interrupted cut necessary to produce the full reaming bit. The bits are machined on the lathe in batches, according to size, and with each resharpening they are reduced in gauge. The capacity of the lathe is about 140 bits per hour, but one operator can easily operate two machines. After gauging, the bits are sharpened in an Oilgear vertical broaching machine (Fig. 4). This consists of four pairs of broach inserts set in a slide. Four bits are held in a worm-geared index, which operates on a sliding table; it automatically carries the bits into the broaching position and simultaneously rotates them by means of an arm, through four positions, 90° apart, until the four cutting edges of the bits have been broached. The machine stops automatically on the completion of the fourth stroke. The capacity of this machine is about 240 bits per hour. A set of broaches will sharpen 500,000 to 550,000 bits, but after 21,000 to 22,000 bits have passed through the cutting edges are sharpened in the shop on a broach grinding machine. This can be done five or six times, then they are returned to the manufacturer to have the cutting teeth ground back to the original depth and width. There is ample stock to permit this to be done at least five times.

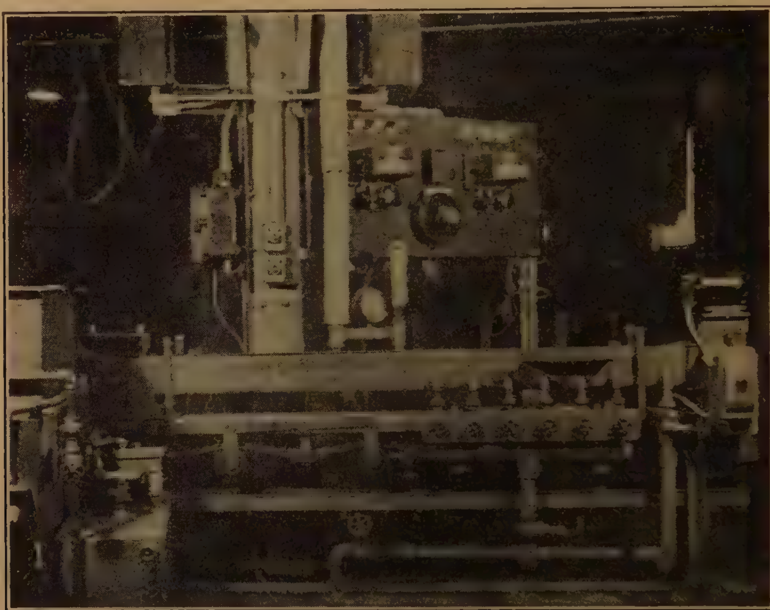

FIG. 5.—BIT CONVEYOR.

Since a heavy cutting oil is used with the turning and broaching operations, the bits are next lowered into a Model V-H 400 Detrex vapor degreaser, and after that are heated to 1425° to 1450°F. in a Walker Products electric pyrometrically controlled furnace. The bits are brought up to required temperature on completion of one revolution of the revolving hearth. They are then placed one at a time on a belt conveyor that is immersed in cold running water to a depth of about ⅜ in. The bits are conveyed in water for about 45 sec. and drop off the end of the conveyor into a pot (Fig. 5). The heating and hardening operation requires the attention of only one man

and he readily turns out 3200 bits in 8 hr. The bits are hardened to 64 to 65 Rockwell C scale. In order to relieve any hardness strains, the smaller sizes of bits are returned to the Lindberg furnace, where they are drawn at 325°F. for one hour.

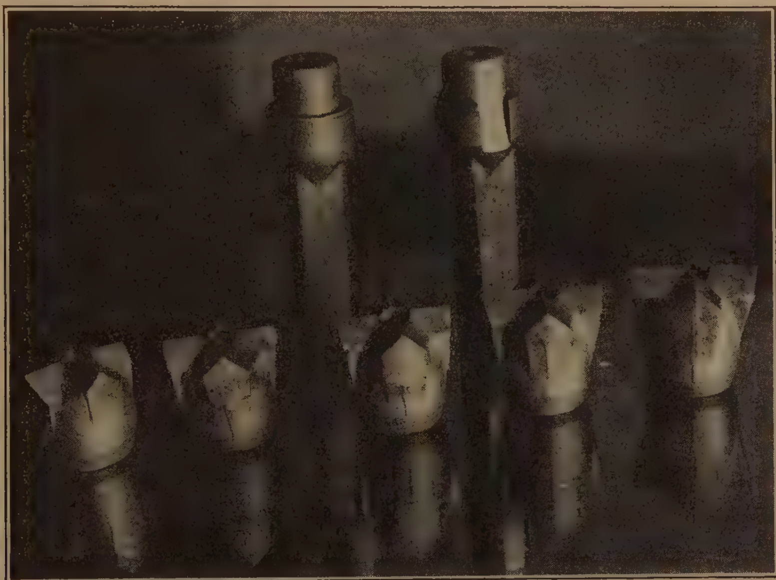

FIG. 6.—INITIAL FORGING, COMPLETED BIT CONNECTION AND FIVE BIT CHANGES.

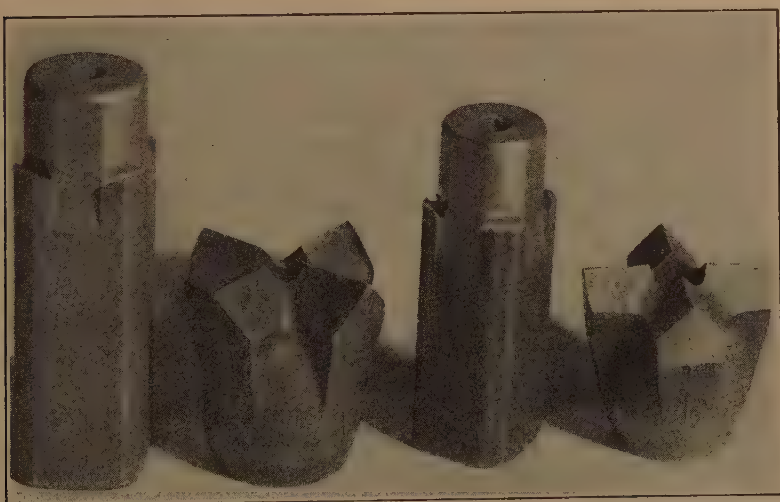

FIG. 7.—PROBABLE FUTURE TYPE OF BIT CONNECTION FOR 1-INCH AND ⅞-INCH DRILL RODS.

FORGING AND SHANKING RODS

The rods are made of 0.78 to 0.83 carbon steel. They are forged in a conventional drill-steel sharpener with a boss or collar and spigot, as shown in Fig. 6, after which they are turned in a lathe to the required

dimensions. This makes the boss and spigot concentric. The flats on the rod are then milled in an adapted Pratt and Whitney Nichols hand miller to conform with the flats of the bit socket. This machine is equipped with a superspacing index head mounted on a swivel base. The index head is indexed for 90° for both the flats and the undercutting of the lips. This makes those operations nearly automatic, after the correct stops for the horizontal and vertical movements of the rod through the cutters have been set. Approximate milling time is about 2 min. Experiments are being made with alloy-steel rods and smaller bit sockets in an attempt to eliminate the collar mentioned above, which is required when 7/8-in. drill rods are used. This change reduces the cost of the rod connection about 3¢, and will permit a further reduction of 1/8 in. in gauge sizes, thus bringing the starter bit down to 1½ in. and the finishing bit to 1¼ in. Failing this, it would seem advisable to adopt 1-in. rods, without the collar as standard (Fig. 7), because the difference in labor between carrying one 7/8-in. rod and carrying a 1-in. rod per shift into a stope is negligible.

Heat-treatment of Bit Rods

The bit end of the detachable-bit rod is preheated to about 800°F. in a muffle furnace. It is then immersed to a depth of about 3 in. in Houghton N.D. liquid heat at 1550°F. for 45 min. and quenched in Houghton No. 2 soluble quenching oil. The hardened end is drawn at 550°F. in Houghton No. 275 draw-temper salts and allowed to cool in the atmosphere. The draw-temper heat overlaps the hardening heat about 3 inches.

The shank end of the rod is heated in an open furnace to 1650°F. about 2 in. back and quenched in Houghton No. 2 soluble quenching oil.

Cost of Plant

The cost of the detachable-bit plant is approximately as shown in Table 1. Duty and sales tax included in these costs amount to approximately 30 per cent on all of the equipment purchased in the United States.

Table 1.—*Cost of Plant*

Oil gear broaching machine.......	$14,000.00
Leblond rapid production lathe...	2,400.00
W. W. Sly shot blast machine....	1,050.00
Lindberg furnace................	1,990.00
Sorting machine.................	1,350.00
Detrex degreaser................	550.00
Holcroft electric furnace.........	4,560.00
Pratt & Whitney milling machine	2,000.00
Broach grinding machine.........	2,100.00
Turret lathe (second hand).......	600.00
Belt conveyor....................	500.00
Labor for installation............	3,300.00
	$34,400.00

The 11 per cent exchange differential effective at present is not included.

Steel-shop Operating Costs

The figures in Table 2 represent the regular sharpening costs for conventional steel and detachable bits for the second period (24 working days) and the year to date, for 1941.

These steel-shop costs are based on a direct labor charge of 68¢ per hour. Total steel-sharpening costs for conventional bits are 5.59¢ as compared to 5.79¢ for detachable bits. However, the cost of new bits, sorting and racking, as well as the cost of bit carriers, extractors and the cost of bit connections on the rods is included in the costs for sharpening detachable bits.

The hardness of the rock from which the performance data of Table 3 are obtained varies greatly. The average penetration for the mine of a 1⅝-in. diameter cross bit with 90° cutting angles, using a standard 3½-in. diameter drifter machine with 90 lb. per sq. in. air pressure at the machine, is about 14 to 15 in. per min. Actually, however, a minimum penetration of 3 to 4 in. per min. and a maximum of 22 to 23 in. per min. are often encountered and must be provided for.

TABLE 2.—*Steel-sharpening Costs*

	Second Period[a]		Year to Date	
	Amount	Per Steel Sharpened	Amount	Per Steel Sharpened
CONVENTIONAL STEEL				
Unit used		138,263		276,084
No. 62 sharpening steel:				
Operating labor	$5,886.28	0.0426	$11,525.93	0.0417
Air-power labor	57.21	0.0004	113.57	0.0004
Steam-heat labor[b]	174.81	0.0012	338.85	0.0013
Total	6,118.30	0.0442	11,978.35	0.0434
3900 gal. oil	507.00	0.0037	1,014.00	0.0037
Air-power stores	445.07	0.0032	895.35	0.0032
Steam-heat stores[b]	305.95	0.0022	479.98	0.0017
Other stores	190.65	0.0014	356.62	0.0013
Total	7,566.47	0.0547	14,724.30	0.0533
No. 62-1 furnace repairs:				
Labor	21.44	0.0002	42.88	0.0002
Stores	3.13		40.45	0.0001
No. 62-2 repair parts:				
Labor	46.97	0.0003	105.88	0.0004
Stores	533.60	0.0039	535.49	0.0019
Grand total	$8,172.09	0.0591	$15,449.00	0.0559
DETACHABLE BITS				
	Number	Amount	Number	Amount
New bits to mine service	10,615	$1,490.03	20,811	$2,928.68
No. 62-4 unit: bits sharpened	39,546		81,358	
Average number times per bit used	4.8		4.9	
	Amount	Unit Cost	Amount	Unit Cost
Sorting and racking	$199.20	0.0049	$ 404.31	0.0050
Sandblast, grading, degreasing	43.22	0.0011	80.74	0.0010
Annealing and drawing	29.46	0.0007	57.30	0.0007
Turning	229.62	0.0058	433.97	0.0053
Broaching	146.06	0.0037	305.52	0.0038
Hardening	45.89	0.0012	113.56	0.0014
Superintendence	85.60	0.0022	196.40	0.0024
Power	86.40	0.0023	172.56	0.0021
Other stores	0.90		0.90	
Total	$866.35	0.0219	$1,765.26	0.0217
DETACHABLE RODS				
Unit: rods shanked	1,130		2,248	
No. 62-5: shanking bit end	$ 141.21	0.1250	$ 268.28	0.1198
Shanking striking end	21.78	0.0193	39.87	0.0178
Hardening and drawing both ends	24.12	0.0213	60.30	0.0267
Total	$ 187.11	0.1656	$ 368.45	0.1637
Unit: bits sharpened plus new bits	50,161		102,169	
Total sharpening and shanking	$1,053.46	0.0210	$2,133.71	0.0209
No. 62-6: Miscellaneous labor	178.58	0.0036	$ 309.16	0.0030
Miscellaneous stores	109.43	0.0021	257.43	0.0025
Hardening No. 1 bits	19.10	0.0004	46.91	0.0005
Total	$ 307.11	0.0061	$ 613.50	0.0060
No. 62-7: Extractors:				
Labor	26.69	0.0005	60.94	0.0006
Stores	27.29	0.0006	70.05	0.0007
Total	$ 53.98	0.0011	$ 130.99	0.0013
No. 62-8 Make carriers:				
Labor	91.26	0.0018	96.19	0.0009
Stores	8.54	0.0002	8.54	0.0001
Total	$ 99.60	0.0020	$ 104.73	0.0010
Grand total, sharpening	$1,514.35	0.0302	$2,982.93	0.0292
New bits to mine service	1,490.03	0.0297	2,928.68	0.0287
Total	$3,004.38	0.0599	$5,911.61	0.0579

[a] Ending Feb. 25, 1941 (24 working days).

[b] As a matter of convenience all steel-shop steam-heating charges are set up against conventional steel. All electric-power charges are set up against detachable bits. This appears to be an equitable distribution.

TABLE 3.—*Performance of Detachable Bits and Rods*

FORTY-SIX 3½-IN. DRIFTERS, 21-R-51 STOPERS USED[a]

	Second Period	Year to Date	Per Actual Machine Shift
Rods damaged or broken..	1,065	2,119	0.86
Bits used..................	51,521	102,611	41.0
Bits lost..................	126	329	0.13
Bits damaged............	267	525	0.21
Holes drilled.............	23,327	47,038	18.5
Feet drilled...............	139,962	282,228	114.0

[a] Miner and helper employed on both types of machines.

SUMMARY

1. Before the present practice, described herein, was adopted many different types of detachable bits were given comparative tests at Hollinger.

2. These tests indicated that no one detachable bit and bit connection adequately fulfills all of the requirements and conditions that are likely to be encountered in mining and tunneling operations.

3. Experiments within the Hollinger organization to evolve a bit and bit connection that would better meet Hollinger requirements resulted in the adoption of the Liddicoat type B bit.

4. Comparative time studies, the assurance from the contract miners that they preferred the type B bit to conventional steel, and the economic potentialities of detachable bits in general (provided the over-all costs per bit use could be kept down to those of conventional steel) offered sufficient inducement for the purchase of the equipment described.

5. Additional sharpening equipment in the amount of $24,000 is needed to meet Hollinger requirements of 7800 to 8000 sharp bits per day. Plant and equipment to sharpen an equal number of conventional bits cost about one-half as much as for detachable bits.

6. Nipping costs for handling approximately 2,400,000 dull drill rods out of underground stopes and development headings and returning the sharpened rods to the stations underground is approximately $58,000 per year, or 2.4¢ per rod.

7. Information derived from the operation of 60 rock drills using detachable bits over a period of 2 years indicates that if the entire mine were using detachable bits the number of rods handled would be reduced to less than 3 per cent.

8. Allowing for an increase of 300 per cent per rod in nipping costs because of the smaller number of rods handled when detachable bits are used, the total nipping costs per year would still be $50,000 less than for conventional steel.

9. The cost of carrying conventional steel rods from the level drill racks up into the stopes is about 1.6¢ per rod. At Hollinger this work is performed by the miner and his helper and is part of their contract. However, probably it is the most fatiguing work required of these men throughout the shift, and since it is performed during the first part of the drilling shift it necessarily reduces their efficiency in their other and more important labors in the stope.

10. Direct labor and material costs of making the type B bit, for which Hollinger at present pays 13¢ per bit, do not exceed 5¢. Since Hollinger would require 500,000 new bits per year if detachable bits became standard practice, a further material saving per bit use, at least for the large consumer of bits, is not an unreasonable assumption.

11. The reduction of drill-rod breakage, upkeep cost of rock drills and consumption of compressed air per foot of hole drilled, because of the better control of the tendency to use dull bits, the greater safety in handling a smaller number of drill rods up and down ladderways, raises and shafts, and the additional hoisting time made available in an already hard-pressed shaft, are all matters favoring the adoption of detachable bits. These, however, are

problems for independent investigation at individual mines.

Acknowledgments

The author wishes to acknowledge his indebtedness to the late Mr. Richard Eddy, formerly foreman in the Hollinger steel shop, and to thank Mr. Percil Liddicoat, formerly assistant foreman, and other shop employees who helped in the development and sharpening practice of the Liddicoat type B bit.

Mr. John Cox, formerly with Thompson Products Co. of Cleveland, Ohio, gave valuable advice on recent machine-tool and electric-furnace developments, and Mr. C. H. Wilkins supplied the pictures that illustrate the article. Dr. W. A. Jones eliminated some inconsistencies and suggested changes in some sentences that were not altogether clear.

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Importance of Falling Ground, Rock, and Coal as an Accident Cause

Report of A.I.M.E. Health and Safety Committee

Edited by John L. Boardman,* Member A.I.M.E.

(New York Meeting, February 1941)

Because of the attention that has recently been given to the health and safety of miners by various organizations such as the A.I.M.E., The American Congress, Mining Section, National Safety Council, as well as the Bureau of Mines and various state mining departments, your committee on health and safety deemed it best to call attention this year to the mine-accident cause that is by far the most important; namely, injuries due to falls of ground, rock, roof and coal.

Each of several members of the committee contributed a brief outline of the importance of this accident cause in his district. While these papers were brief, it would require too much time and space to give them in full, therefore we present an outline of each and summation of all.

Report of M. D. Cooper, Hillman Coal and Coke Co.

M. D. Cooper, Division superintendent, Hillman Coal and Coke Co., Pittsburgh, Pa., contributed the following:

During the period from 1930 to 1939, accidents from falls of roof in bituminous mines average 54.3 per cent of all accidents. In mechanized mining, there is a tendency toward reduction in this class of accidents because the operator of the machine is usually at a distance of 10 to 20 ft. from the face, whereas in hand loading the miner is close to the face during most of the shift. No doubt the use of aluminum and light steel beams supported at regular intervals by jacks, a common practice in mechanical mining, has contributed to the favorable record so far developed.

Report of M. H. Fies, De Bardeleben Coal Corporation

Milton H. Fies, Vice-President, DeBardeleben Coal Corporation, Birmingham, Ala., says:

With reference to the types of mining, it is suggested that a clear understanding be had as to the character and thickness of the directly overlying strata, with especial consideration to the strength and in some cases an analysis of it. In coal mining, generally speaking, overlying strata higher in silica than others is stronger.

In changing from narrow to wide places, the procedure should be experimental, both as to the width and depth of places and as to the nature of the necessary timbering. Since most of the accidents in coal mining particularly are due to falls of roof and coal, this phase of mining should be attacked with vigor

There is no substitute for discipline and, in this connection, under present conditions it is important that cooperation by the labor organizations be sought and that a spirit of fairness prevail.

Frequent and thorough inspection of all active working places should be made.

Manuscript received at the office of the Institute Dec. 19, 1940. Issued in Mining Technology, September 1941.

* Chairman, Bureau of Safety, Anaconda Copper Mining Co., Butte, Mont.

Experience has taught within the last three or four years that increased supervision, viz., the number of foremen, should give better results.

Complete investigations of all types—serious, fatal and nonfatal accidents—should be made and the causes determined.

Picture films and radio programs have been found serviceable in connection with safety education.

Where changes in safety rules are contemplated, it is well to have the men themselves take a part in the formulation of such changes, which in itself is education.

Report of E. E. Hunner, M. A. Hanna Co.

Earl E. Hunner, General Manager, Lake Superior Iron Mines, M. A. Hanna Co., shows that:

In 1920 there were 42 disabling falls of ground for each million gross tons of iron ore produced, and in 1935 only eight per million tons. Another company shows the fatality record listed in Table 1 for the eight-year period prior to 1911 and for the eight-year period prior to 1939.

Table 1.—*Record of One Company*

Cause	Number of Fatalities	
	Early Period	Recent Period
Falls of ground	27	1
Shaft accidents	18	0
Explosives	13	0
Haulage	8	0
Falls of persons	4	0
Other causes	1	2
Total	71	3

By far the largest part of underground ore mined by the slicing system wholly on the Mesabi Range comes from St. Louis County, whose mine inspectors' reports the past five years show the fatalities listed in Table 2.

Certain common-sense rules must always be followed, such as never permitting miners to reenter a working place after firing until the smoke has cleared out and good illumination is provided. Barring down for loose material must be the first step followed by well-proven timbering methods, using only the best of timber, and often H-beams or other type of steel supports above cap timbers, followed by proper forepoling and tight blocking of timbers and open spaces.

Table 2.—*Data from Mine-inspectors' Reports, St. Louis County, Minnesota*

Year	Fatalities		Serious Injuries	
	Total	Falls of Ground	Total	Falls of Ground
1935	3	3	5	3
1936	4	0	6	3
1937	0	0	20	5
1938	1	1	9	3
1939	2	1	8	4
Total	10	5	48	18
Percentage		50		38.5

An example of conditions creating fall of ground accidents is found in the case of Mesabi Range top slicing, where the top slice is drawn from under the overlying glacial overburden containing enormous granite boulders, which produce a concentrated load above the timbering not revealed until a cave-in occurs. In some instances the top slice may be under a slate capping, hard enough to overhang after the room is sliced out and blasted in, falling at some later time when the rock arch weakens and shears, crushing in the open portion of the new room adjoining the ore pillar and endangering the miners.

In parts of the Michigan ranges, the iron ore is found in folded areas of black slate and it is necessary to leave about 6 ft. of ore to keep the slate in place. When the black slate is exposed in a stope, it often crumbles and the supporting walls undermine. Small mine fires, due to the occurrence of sulphur in the loose, crumbling slate may occur, which will necessitate the

bulkheading of all entrances to the stope to cut off the oxygen supply and also keep the fumes from the burning slate from penetrating other working places. The intrusion of black-slate seams or small dikes in the iron formation makes a dangerous condition. If the seams are hidden in the bench or back of slice, the chances are that a block of ground might become loosened and fall without warning.

Close supervision is the greatest asset in the prevention of "falls of ground." In open-stope mining a foreman must have a mental picture at all times of the relative position of one gang to another, judge the ground properly as to width of bench and overhang, examine bottom of bench for fractures and sloughing, and when mining operations are approaching limits where black slate might be encountered, keep a 10-ft. test hole drilled ahead to prevent blasting into and exposure to the air of the slate wall.

The National Safety Council meeting in Chicago in October 1940 stressed the importance of making each workman part of the safety program by allowing him to express his views freely if they were of a constructive nature. This system has been tried over a period of years in the Lake Superior district, with excellent results.

In organizing for group meetings with the men, three employees, of whom one is named chairman, take the responsibility for making accident prevention predominate for one month. Toward the latter part of the month, these three men make a complete inspection of the property, and following the lunch hour on a given day, the chairman calls a safety meeting with all workmen on the shifts in attendance. The report of the inspection committee is then read and discussed, and if no comments are made the suggestions stand approved and corrections are made without delay.

After the committee report has been accepted, the meeting is then open for any constructive ideas from the employees, and, it might be stated, many are received. The safety engineer then gives statistical information and a résumé of happenings in the district. Before the meeting adjourns, the committee appoints three of the crew to carry the program through the following month. All safety meetings are held on company time and usually take one hour.

Report of H. J. Mutz, International Nickel Company of Canada

In Canada, Herman J. Mutz, Superintendent of Mines, International Nickel Co., Copper Cliff, Ont., Canada, presents the following very interesting and useful information:

Accident-classification figures for 1939 released by the Ontario Mining Assn. show that falling ground still holds its place as the greatest accident hazard. During the year no less than 15 per cent of all the compensable accidents and 41 per cent of all fatalities occurring underground in all Ontario mines were caused by falling ground.

Statistics for the producing mines of the province, numbering about 65, with an aggregate of about 29,000 employees, show that over the past three years rock falls have averaged 15.3 per cent of all accidents and 30.2 per cent of all fatalities. In 1939 the corresponding figures were 15.6 per cent and 32 per cent. While these showed a decrease over 1938, they are still above the three-year average.

The frequency of this type of accident per 1000 men employed shows a decided improvement in 1939 over the three-year average, from 8.35 to 8.00, while the fatal frequency dropped from 0.39 to 0.31.

A general classification of accidents caused by falling ground in producing mines of Ontario is given in Table 3.

A further analysis shows that 65 per cent of the accidents over the three-year period have been suffered by drillers and drill helpers and 15 per cent by timbermen, and that 50 per cent of the accidents under

this heading have resulted in injury to feet and legs. These facts stress the crying need for greater control by the supervisors of the men who are permitted to attempt the dangerous task of scaling; more protection in the way of head cover and the insistence on the use of protective equipment for the feet, such as safety shoes and spats.

the back with Leyner drills is being superseded by long-hole diamond drilling. This makes possible the drilling of a complete stope back from a protected area, such as a slot against the pillar; and in some cases this drilling is done from within the pillars.

Education of miners is of primary importance in avoiding accidents from falling ground. Supervision plays an important part also. Stress is placed on the accident-prevention phase of supervision. Shift bosses and foremen are trained to maintain a continued and constant watch for carelessness.

TABLE 3.—*Accidents Caused by Falling Ground, Ontario*

Cause	1937		1938		1939		Three-year Average	
	Number	Fatal	Number of Accidents	Fatal	Number of Accidents	Fatal	Number of Accidents	Fatal
While scaling or timbering	95	2	78	2	97	0	90	1.3
While setting up or drilling	53	0	62	0	51	2	55	0.7
Loose ground (not otherwise classified)	95	6	106	16	94	7	98	9.6
Totals	243	8	246	18	242	9	243	11.6

The type or method of mining has had a definite influence on the number of accidents from falls of ground. The general attitude among mine operators of the province is to change to methods that provide more back support as their experience shows that occasional falls of ground due to carelessness can be expected. The change usually is made from cut and fill to a method of square setting. Standard practices are then provided for all to follow. Booming out ahead of square sets before standing new timber is customary practice.

It is strikingly noticeable that in one camp where square-set stoping is standard practice, the frequency of accidents from falling ground in 1939 was 3.03 per 1000 men employed as compared to the general rate for the province of 8.00 per 1000 men employed. The fatal accident rate was 0.25 in the camp mentioned as against a general average for the province of 0.31.

A change is being made to more protective methods of mining such as square-setting instead of cut-and-fill mining. Where shrinkage stoping has been employed, the usual method of drilling off

The education of miners and other stope employees is perhaps the most important step taken in the prevention of accidents from falling ground.

Various forms of educational schemes are in use in the different mining camps, chief among them being the following:

Standard practices or codes have been developed for all phases of an underground operation. These codes contain detailed instruction for all classes of employees and cover every part of the different types of work from the time the men enter the mine until their return to surface. These codes form a basis on which the supervision can criticize the manner in which work is being performed. All classifications of men are thoroughly instructed in the codes pertaining to their type of work.

Foremen and bosses are used as instructors in regular scheduled classes on mine

standard practice. This instruction is carried on even after men have passed initial school tests. The classes are held in a lecture room provided on surface at the mine plant. In some cases the "lecture rooms" have become a standard requirement in all new plant layouts.

Stope schools are used in some cases to train all new employees. The inexperienced men are sent to the stope school for an instruction period. They receive instruction in safety methods of a particular operation and are taught proper methods of working. These new men are judged also on the basis of adaptability to underground work. They are usually "graduated" from the school stope as muckers and helpers.

After a six-months period at regular work the most promising types are returned to the school for instruction in drilling, scaling and timbering. In this way the intelligent, dependable types are selected as key men. This feature alone is in a large way responsible for reducing falling-ground accidents wherever this scheme of education is employed.

Safety classes are held, the instructor being the mine safety engineer.

Moving pictures and posters are used extensively. These depict the right and wrong ways of handling bad ground and also show the results of the incorrect method or of carelessness.

The apprenticeship scheme is used to some extent. Men are not allowed too rapid an advance. They are advanced through the different classes of work as they show proficiency and knowledge. This advancement is controlled by the mine safety engineer, who conducts regular tests of all men proposed for advancement.

Report of G. B. Pryde, Union Pacific Coal Co.

In the bituminous coal industry, G. B. Pryde, Vice-president in charge of operations, Union Pacific Coal Co., makes the following report:

Speaking for the coal mines in the state of Wyoming (and from my study of this question at different times, I feel that this will hold good for the Rocky Mountain region generally), for the six-year period 1934 to 1939, inclusive, in all wyoming coal mines, while accidents from all sources showed a substantial reduction, those caused by falls of rocks and coal did not show a comparable diminution. Table 4 shows the figures.

TABLE 4.—*Accidents in Wyoming Coal Mines*

Year	Accidents from Falls of Rock and Coal	All Accidents	Percentage of Accidents from Falls of Rock and Coal to All Accidents
1934	73	231	31.6
1935	77	259	29.7
1936	86	279	30.8
1937	68	250	27.2
1938	61	172	35.5
1939	66	178	37.1

The situation with one of the largest operators in Wyoming is about the same as for the state as a whole. Table 5 shows the results obtained in that company.

TABLE 5.—*Statistics from One Operator in Wyoming*

Year	Accidents from Falls of Rock and Coal	All Accidents	Percentage of Accidents from Falls of Rock and Coal to All Accidents
1934	19	59	32.2
1935	17	63	27.0
1936	19	53	35.8
1937	14	40	35.0
1938	10	31	32.3
1939	14	27	51.9

Accidents from all causes were more than cut in half, although the reduction in accidents from falls of rock and coal was very disappointing.

One reason for the large number of accidents of this kind in the Rocky Mountain

region is the fact that mining is done on pitching seams; many accidents occur from rolling coal and rock because of the heavy pitch. Mine operations are carried on with pitches of from 4 to 25 per cent. Many of these accidents affect the legs. In other words, underground employees are more susceptible to accidents of this kind on pitches than on flat seams.

The measures being taken to prevent injuries from this source are the introduction of hard hats and hard-toed shoes, which are each 100 per cent in the Union Pacific mines. These two things alone have been responsible for a great diminution in accidents from falls. Shin guards have been tried, to protect against leg injuries on account of rolling coal and rocks, but their introduction has not been very successful. The men object to using them, and so far they have not been found practical. I understand they are greatly used in Great Britain and in some parts of continental Europe. Systematic timbering has been of material benefit in reducing accidents from this source.

Report of C. M. Fellman, Montreal Mining Co

C. M. Fellman, Safety Engineer, Montreal Mining Co., Montreal, Wis., contributed the following information on the subject from the iron mines of the Lake Superior district:

The methods of mining in their order of importance as used in the district are top slicing, sublevel caving and open stoping. Table 6, accident rates compiled by the U. S. Bureau of Mines, shows that top slicing and sublevel caving, methods under which most of the iron ore coming from underground operations is produced, are the safest.

First, serious consideration should be given to the selection of captains and shift bosses who are above the average in intelligence, friendly in disposition, cool thinkers under stress, and who understand that safety in mining is as important as the production of ore.

Table 6.—*Accident Rates in the United States*
U. S. Bureau of Mines

Method of Mining	Rate per Million Man-hours Worked	
	Killed	Injured
Top slicing	0.45	20
Sublevel caving	0.63	25
Square set	1.28	133
Open stopes	1.36	80
Cut and fill	1.63	103
Shrinkage stopes	2.25	122
Block caving	2.73	143

It is important to have the working place well illuminated, so as to be able to make a visual inspection of the back and sides at all times. Proper flood lamps should be provided to permit the miners to inspect the back and sides from a safer distance than is possible with the miner's electric cap lamp or carbide lamp.

The timber used should be the best obtainable and of proper size and length for the size slices and openings used in each variation of the top-slicing or sublevel caving methods employed in various types of ore bodies.

The miner should not reenter his working place after blasting until the smoke has been cleared out. Many a man has been injured by a fall of ground because he could not see the back clearly because of smoke.

A miner should, of course, sound the ground and bar down the loose carefully when entering the breast after a blast. While standing under the last cap, he will bar down all loose slabs and chunks from the back, breast and sides that he can see and handle from this position, before moving in under the newly exposed back to continue trimming.

Probably experienced miners, who have been trained in the school of practical experience, are the most important factors in preventing accidents from falls of ground.

The value of experience is shown by the great reduction in accidents in recent years, as compared with the early days when many so-called "greenhorns" were employed in the mines.

Report of Martin Byrnes, Anaconda Copper Mining Co.

Martin Byrnes, of the Anaconda Copper Mining Company's Bureau of Safety, Butte, Mont., contributed the following:

In the Butte operations of the Anaconda Copper Mining Co., the safety department compiles a monthly report covering all accidents. Records for the past 20 years show that falls of ground and rocks have been responsible for 45 per cent of the fatal accidents, 33 per cent of the serious, and 33.5 per cent of the total accidents.

The influence of mining methods upon the accident rate from falling ground is shown in Table 7.

Table 7.—*Injury Rates from Falling Ground and Rocks, Anaconda*, 1939

Mining Method	Fatal	Serious	Slight	Total
Per Million Man-hours Actually Worked in Various Types of Stopes				
Timber rill stopes.....	0.00	77.62	41.75	129.37
Open rill stopes.......	6.88	66.75	43.75	117.37
Horizontal cut-and-fill stopes.............	3.25	52.37	55.62	111.25
Square-set stopes......	2.75	55.12	18.75	76.62
Stull stopes...........	7.80	47.50	7.25	75.12
Per Million Man-hours Based upon Total Hours Worked				
	0.87	19.2	10.6	30.67

It has been recognized for years by the operating department that the shift boss is the key man in the accident-prevention program. Every effort is made to select bosses who have a wide experience in all phases of modern mining methods, the ability to recognize the hazards and to instruct their crews to handle them without injury. It is understood that if a boss must spend the greater part of his time traveling from one working place to another, with a minimum of time actually spent in the working place, he cannot give the close supervision necessary for efficient mining and proper safety measures. Consequently, the size of the bosses' crews has been reduced to an average of 23 men. To keep up the bosses' interest in and knowledge of mine safety, information circulars from the safety department are distributed at the mines at least twice a week. These circulars describe accidents that have occurred at various mines and give an accurate description of conditions that caused the accident and a reminder to watch out for similar conditions on other levels.

In an effort to secure the cooperation and good will of the men and to sell them personally on safety, a regular schedule of short lectures was instituted a year ago. Each crew attends one lecture a month on company time. The talks on safety are varied by showing motion pictures made by the safety department in Butte, showing typical underground conditions in the Butte mines and, of course, the correct way to handle a variety of conditions. A slide projector also is used to illustrate the talks on such subjects as falling ground, proper placement of stulls, and proper blocking of timber. It has been found that an illustrated talk holds the attention and interest of the men better than one without slides.

Supplementing the efforts of the bosses and safety engineers, there are large lighted bulletin boards showing the number of accidents for the year and the number of shifts worked by each crew since the last accident. The list is arranged in descending order of merit. The boards are well lighted and placed near the headframe, where the men can read them while waiting to be lowered into the mine. Every effort is made to encourage a competitive spirit among the bosses and men to have their crew top the board.

Summary

Records of the Bureau of Mines indicate that 13,189 miners have been killed by explosions of gas and coal dust in the coal mines of this country during the past 50 years, but during the same period in these mines 46,476 men have lost their lives by falls of roof, sides or gob. Thus, the hazard to life from falls of rock or coal is more than 3.5 times as great as that from explosions of gas and dust. In the metal mines the proportion of total fatalities due to falls of rock is very nearly the same as for coal mines.

Analysis of the reports quoted in the preceding pages does not reveal that the type of mining is the determining factor in the causes of such accidents, but there is a pronounced preponderance of opinion that these accidents are preventable by the proper training of miners and the sincere efforts of mine supervision.

Miners must be taught the threefold purposes of using timber; *i.e.*, to provide staging, to support loose rocks and to provide a warning when rock begins to move. They must be impressed with the unreliability of their sense of hearing as a means of detecting loose ground and must also use the senses of touch and sight to the fullest possible extent.

It is a well-recognized fact that very few fatal injuries in mines are caused by caves. Nearly all of them, in fact, are due to being caught by falls of comparatively small pieces—such pieces of coal or rock as might weigh from a few pounds to a ton or more, and nearly all of which could either be taken down or blocked up safely by the use of timber. It must be remembered that for the past 50 years in the coal and metal mines in this country about 1000 lives per year have been taken by this cause. If the killing of 231 miners per year by the more spectacular explosions of gas and coal warrants the tremendous attention it is receiving, and we all agree that it does, certainly the prevention of 1000 deaths from this equally controllable hazard of falling ground, rocks and coal challenges the ingenuity of the supervisory forces of all our mines.

Teaching Design in Mining Engineering Curricula

By J. W. Stewart,* Member A.I.M.E.

(New York Meeting, February 1941)

The aim of this paper is to point out the various ways in which design is taught in standard four-year mining engineering curricula in American colleges and universities; to discuss the reasons apparently justifying the great diversity of procedure that exists along this line; and to indicate possible solutions to the difficulties now attending the teaching of design.

A study was made of typical four-year mining engineering curricula in 15 colleges and universities of the United States (Table 1). Great diversity in content of the various design courses, if taught as described, became apparent immediately. It appeared that it might be both interesting and instructive if the cause for this colorful variety could be found. Was the cause local mining conditions, departmental individuality, or actual necessity?

Study of the school catalogues revealed that the various design courses could be classified into two fairly distinct types; namely, (1) those dealing with mine equipment or structures, and (2) those dealing with underground layout of mine workings. It seemed reasonable to label the first type of course "mining design" and the second type "mine design." Some such distinction must have been in the minds of mining educators, for we find both names (i.e., mine design and mining design) used to designate the design courses included in the various mining curricula. Unhappily, the name and the apparent content of such courses do not, in all instances, conform to the classification made here, although in most instances such conformation does obtain (Table 1). It is possible that in the exceptional case the method of teaching a design course, and consequently the content of it, may have been changed at some time without change in the name of the course.

Apparently only 3 of the 15 schools made an effort to cover both types of design in their curricula (Table 1). Two other schools did not list or describe any course of study that could with certainty be taken as a design course. Of the remaining 10 schools, 9 gave a mining-design type of course and one gave a strictly mine-design type. Obviously, the preponderance of design courses afforded are of the mining-design type. Why? To answer this question, it is necessary to make a study of the prerequisites for the two types of design courses taught. In all schools the design courses were taught in the senior year; in eight, design was taught throughout both semesters; in two, it was taught during the first semester only; and in three it was taught during the second semester only. Consequently, excepting the last three cases noted, it is obvious that all prerequisites for a design course should be completed by the end of the junior year.

Prerequisites

What are the probable prerequisites for the different kinds of design courses as they appear to be taught? Let us take first the case of the mine-design type of course, having to do with layout of underground workings. Here it would seem that logical

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TABLE 1.—*Study of Four-year Mining Engineering Curricula*

School No.	Mine Surveying	Elements or Principles of Mining	Mining Methods or Systems	Mine Equipment (Compressed Air, Haulage, Hoisting, Pumping, Ventilation, Etc.)	Design			Ore Dressing	Coal Preparation	First Mining Course
					Catalogue Name	Content of Course	Remarks			
1	Jr. 2[a]	Jr. 1	Jr. 2	Sr. 1, 2	MNG.[b]	MNG. Sr. 1	Structures and equipment	Sr. 1	Sr. 2	Jr. 1
2	So. 2	Jr. 1	Jr. 2	Jr. 2, Sr. 1	M.D.[c]	M.D. and MNG. Sr. 1 and 2	Structures and equipment	Sr. 2	Sr. 2	So. 1
3	Jr. 1	Jr. 1	Jr. 2	Sr. 2	MNG.	MNG. Sr. 1 and 2	Equipment	Sr. 1, 2	Sr. 1, 2	Jr. 1
4	Jr. 2	So. 1	Sr. 1	Sr. 1	M.D.	M.D. Sr. 1 and 2	Underground	Sr. 2	Sr. 2	So. 1
5	Sr. 2	Jr. 1	Jr. 1	Jr. 2	MNG.	MNG. Sr. 1 and 2	Structures	Sr. 1	Sr. 1	Jr. 1
6	Sr. 1	So. 2	Jr. 1	Sr. 2			Apparently no design	Jr. 2, Sr. 1		So. 2
7	Jr. 2	Jr. 1	Jr. 1	Jr. 1, 2	Mng. Const.	MNG. Sr. 1	Construction practices	Jr. 1	Jr. 1	Jr. 1
8	Jr. 2	So. 1	Jr. 1, 2	Sr. 1, 2	M.D.	MNG. Sr. 1 and 2	Structures	Sr. 1	Sr. 2	So. 1
9	Jr. 1	Jr. 1	Sr. 2	Sr. 1			Apparently no design	Jr. 2, Sr. 1		Jr. 1
10	Jr. 1	Jr. 1	Sr. 1	Sr. 1, 2	C.E.	MNG. Sr. 2	C.E. dept. (stresses)	Sr. 1, 2		Jr. 1
11	So. 1, 2	So. 2	Jr. 1, 2	Jr. 1, 2	M.D.	M.D. and MNG. Sr. 1 and 2	Equipment and underground	Sr. 2	Sr. 1	So. 1
12	Jr. 2	So. 2	Jr. 1	Sr. 1	MNG.	MNG. Sr. 2	Structures and equipment	Sr. 1	Sr. 1	So. 2
13	So. 2	Jr. 2	Sr. 1	Sr. 1	M.D.	M.D. and MNG. Sr. 2	Equipment and underground	Sr. 1, 2	Sr. 1, 2	So. 1
14	So. 2	So. 2	Jr. 2	Jr. 1, 2	Engng. Const.	MNG. Sr. 1 and 2	Structures	Jr. 1, 2 Sr. 1		So. 2
15	Jr. 2	Jr. 1	Jr. 1, 2	Sr. 1, 2	MNG.	MNG. Sr. 1 and 2	Equipment	Sr. 1, 2		Jr. 1
Summaries...	So. = 4 Jr. = 9 Sr. = 2	So. = 6 Jr. = 9	Jr. = 11 Sr. = 4	Jr. = 4 Sr. 1 = 5 Sr. 2 = 6		M.D. = 1 MNG. = 9 M.D. & MNG. = 3 NONE = 2		Jr. 1 = 1 Sr. 1 = 7 Sr. 2 = 7	Jr. 1 = 1 Sr. 1 = 3 Sr. 2 = 6	So. 1 = 5 So. 2 = 3 Jr. 1 = 7

[a] The number following So., Jr. or Sr. indicates the semester.
[b] MNG. = mining design.
[c] M.D. = mine design.

prerequisites are: (1) sound courses covering mine surveying; (2) elements (or principles) of mining (including prospecting and exploration, drilling and blasting, development, etc.); (3) mining methods (or systems); and (4), in a personal opinion, a good knowledge of barodynamics—the branch of mechanics dealing with rock strengths and movements. Observation indicates that in only a very few schools is sufficient attention paid to the last of these prerequisites, although in general the courses in mechanics that are given are completed by the end of the junior year. In considering the other prerequisites mentioned, it appears that in the great majority of cases these are fairly satisfactorily met by the end of the junior year (Table 1).

Then why is it that only one school endeavors to give a strictly mine-design type of course, and only three more make an attempt to give a combined mining and mine-design course? The answer probably lies in the inherent lack of quantitativeness encountered in attempting to teach mine design without the use of barodynamics. Without such a tool it appears that there is too much rule-of-thumb, guesswork, and borrowing in determining the proper mining system to employ, and in deciding upon such items as the proper size of pillar, stope, room, etc., to be used under a given set of conditions. This, of course, is a personal opinion based on individual experience in teaching design.

Perhaps, then, it is not surprising to find the great majority of instructors resorting to a mining-design course of either the equipment or structures type. But even here it does not appear that an instructor has complete leeway in his choice, largely because of essential prerequisites. Let us look again into this matter of prerequisites as it concerns us here.

First, consider the equipment type of mining-design course. The essential prerequisite here is a thorough course covering the equipment used in such operations as air compression, haulage, hoisting, pumping, and possibly ventilation. The study of mining curricula reveals that only 4 out of 15 schools complete this course during the junior year—in time to provide a full year for an equipment-design course. Further, only 5 of the remaining 11 schools complete such an equipment course by the end of the first semester in the senior year, making it more difficult to teach an adequate design course of this type. The remaining six schools do not complete the prerequisite course until the end of the senior year, making it difficult or impossible to teach an equipment-design course in the senior year.

Under this handicap it is not surprising to find that the majority of schools resort in whole or in part to a course involving the design of mine structures. Such a course might cover the design of headframes, ore bins, shop buildings, etc. The essential prerequisite for such a course would be college mechanics, and it is usual in almost all engineering colleges to complete this course by the end of the junior year, thus making it readily possible to teach a fairly satisfactory mine-structures design course in the senior year. This actually seems to be the favorite type of design course, as study of the curricula of the 15 colleges shows. It appears from Table 1 that at least 8 out of the 13 schools giving a design course definitely give a structural-design type of course, in whole or in part.

Another type of mining-design course, not strictly mining and consequently not mentioned heretofore, is one based on the design of an ore-dressing mill or coal-preparation plant. But again the average instructor has a definite obstacle placed in his way with regard to prerequisites. Study of Table 1 shows that only one of the schools listed completes the prerequisite courses in ore dressing or coal preparation by the end of the junior year, thus making

possible a full-year design course. Of the remaining schools only about half complete their undergraduate courses in ore dressing by the end of the first semester of the senior year, thus making it just possible to teach an ore-dressing design course during the second semester of the senior year. The situation is still more critical if a coal-preparation design course is contemplated, for only about one-third of the schools involved complete the prerequisite course in coal preparation by the end of the first semester of the senior year. The other schools do not complete their beneficiation courses, either coal or ore, until the end of the senior year, obviously making it impossible to teach an undergraduate design course of this type. As a matter of fact, only one school made any reference to beneficiation-plant design in the catalogue descriptions noted.

Thus, to answer the earlier query, it would seem largely to be a matter of actual necessity, owing chiefly to prerequisite requirements, that determines the type of design course that is to be taught at a certain school. At first thought one might be led to believe that the difficulty with prerequisites may definitely be bound up with the time at which the student is given his first mining course. This, however, does not appear to be true (Table 1). About half of the schools start their students on their first mining course in the sophomore year, while the other half do not start their students until the junior year, yet there does not seem to be any correlation between this starting time and the progress made by the students upon arrival at the senior year, when design most likely will be taught. A comparison of the last column in Table 1 with the remainder of the table will reveal this fact. As we all know, there are other essential engineering courses, not mining in nature, that must be taught in the interim between the sophomore and senior years. These courses definitely interfere with the teaching of junior mining courses, and this difficulty constitutes a serious problem in the formulation of a mining engineering curriculum.

Conclusion

It appears that considerable difficulty attends the teaching of a comprehensive design course in the present standard four-year mining engineering curriculum. It is admitted that a fair case can be made for teaching a structural type of mining design in practically all schools, although such a course must necessarily be limited in scope. Also, certain schools are in a fairly favorable position to teach an equipment type of design course, but too many schools are deficient in this respect to make it possible to say that all schools generally can do this. Certain it is that few schools are in a favorable position to teach a really scientific and quantitative underground mine-design course. Too few of our students, and teachers, have the necessary prerequisites for pursuing such a course with beneficial results. However, the time may shortly be here when such a scientific underground-design course may possibly be taught. Studies being conducted, particularly those at Columbia University School of Mines, are definitely advancing the horizon in the field of barodynamics, a necessary tool for the prosecution of such a course. Even so, we will then be confronted with the acute problem of placing still another prerequisite course, albeit an essential one, along with an ever-growing list of such courses, in an already overcrowded four-year curriculum.

What possible solutions are available in these present difficulties? A few are indicated as follows, although some may be thought of as undesirable or impracticable:

1. Rearrangement of the content of the present four-year mining engineering curriculum, so that the prerequisites for a design course be given preference, and that other courses such as the humanities, economics, etc., which do not require

prerequisites, come last. This solution is a practicable one, but may be thought of by many as undesirable from the point of view of lessening emphasis on nonengineering courses. As a matter of fact, many engineering authorities assert that more nonengineering courses than are given at present should be included in a well-rounded engineering curriculum.

2. Provide specialized options or elective courses in the last two years of the mining curriculum in order that students may specialize more particularly in such fields as mining, mining geology or ore dressing. This would tend to lessen the large number of prerequisite courses now called for in the present all-inclusive four-year curriculum, and permit the teaching of certain particularized prerequisites (barodynamics, for example) that would enable this specialization to be accomplished successfully. This is a practicable solution, but may be objected to on the ground that further narrowing in the training of mining engineering students by specialization is undesirable. Also, practically, such a procedure might tend seriously to handicap schools with small enrollments.

3. Increase the present four-year mining engineering curriculum to one of five years' duration, either with or without specialization during the last two years. In either case, better preparation is made possible for teaching the more advanced mining courses, including design. At the same time opportunity would be provided for the inclusion of more advanced service courses in geology, electricity or heat engineering—fields that have advanced so extensively in recent years that present service courses scarcely more than touch upon them. Also, it would be possible to include more of the humanities, thus rounding out the curriculum in a manner recommended by many leading engineering authorities.

This solution might be opposed on the ground that it would tend to work to the disadvantage of mining education, unless all engineering curricula were increased to five years uniformly. However, it might be noted in this connection that every new move must be started by someone. Others might object to devoting more than four years to any engineering curriculum on the ground that no more than four years are needed to cover the essentials of the subject matter. In the light of present-day extensive scientific and engineering developments, the validity of this objection may be questioned. The modern engineering profession is fully as complex as either the medical or legal profession, and comparatively speaking a five-year period is not too long to devote to academic engineering training.

4. Teach design as a graduate course, given subsequent to completion of the present four-year mining curriculum. All the prerequisites necessary to the successful prosecution of a comprehensive design course could then be met satisfactorily in the regular four-year undergraduate curriculum. This procedure would largely eliminate difficulties that at present attend the teaching of design in undergraduate mining curricula.

This solution may be objected to on the ground that few of our mining students will take graduate work, and that all should have the benefit of some sort of design course before going into industry. This does not seem to be the universal opinion of educators, for we find that some of our mining schools do not give a design course of any sort. This may not be so serious as it might seem at first glance, for, after all, these recent graduates will not be called upon to do any extensive design for several years after leaving college. Those who later may see that they are going to be called upon to particularize in a special field of design may well return to school for a period of graduate study conducted along lines of their needs. This often is done in other well-developed fields of engineering.

DISCUSSION

D. L. McElroy,* Morgantown, W. Va.—Prerequisites are the main difficulty to setting up all types of mine-design courses, except possibly mine structure. For that, fundamental courses in mechanics are the prerequisites, which certainly are taught in all accredited schools.

I feel that better results can be obtained by combining the mine-design course and the mine-equipment type of design course. The reason for this feeling is the fact that mine projection, methods of working and such material must be correlated with the mine equipment used. Neither of these two phases of mining can be designed without close correlation with the other.

* Director, School of Mines, West Virginia University.

Classroom instruction is probably the most important use of barodynamics at the present, but it should be tempered with results obtained by its application in practice. At our school, for example, this subject has been discussed with the staff of the mechanics department in order to have barodynamics included as a part of the course in strength of materials.

The solutions of the problem of prerequisite requirements for courses in mine design, as outlined by Professor Stewart, pretty well exhaust all possibilities. It is, therefore, the problem of each teacher to accept this responsibility and do what his judgment dictates as best. In arriving at this final arrangement of curriculum, the teacher should keep in mind the field of work into which most of his students go. In other words, the teacher must first decide the purpose of this course.

Post-collegiate Education of Mining Engineers

By Thomas T. Read,* Member A.I.M.E.

(New York Meeting, February 1941)

Mining, which is at least twenty centuries old, was at first, and long, wholly a practical art. Little more than two centuries have elapsed since the inception of the idea that general education and a knowledge of natural science would be of advantage to those engaged in the mining industry. Though schools of mining began to develop in Europe early in the eighteenth century the concept of educational preparation for technical conduct of the industry was slow in winning general acceptance in the industry itself. John Hays Hammond, in his autobiography,[1] recalls that when he sought employment with a mining company after he had graduated from Yale and Freiberg he had to assure its executive, before being even considered for employment, that he did not consider that he had learned anything about mining. This was as late as 1880.

Existence in the industry of a prejudice against education of the academic type is perhaps an explanation why inadequate provision has been made by the mining industry for the continuing education of technical employees after they enter employment, although engineering education prior to engaging in industry has now won general acceptance. I am familiar with the existence of "loop" courses for engineering graduates in metallurgical plants, of schemes for giving mining graduates "practical experience," and of the interest in vocational courses for workmen which sprang up in the period of labor shortage during and immediately following World War I (but which quickly died when the depression of 1921 again made experienced workmen available in sufficient supply), having been chairman of a committee appointed by the Mineral Industry Education Division to make a survey of existing provisions for post-employment training for engineers in our industry. The results of that survey, reported elsewhere, clearly indicate that it is inadequate as to amount; the aim of this paper is to present evidence for belief that it is also inadequate in quality.

To avoid becoming too involved, no reference will be made here to post-employment vocational training for workmen, which is a subject differing so much from post-employment education for engineers as to require separate treatment. Neither will postgraduate education for metallurgical engineers be discussed, because a distinction must then be made between training for research work and for plant operation, whereas mining engineers engage almost exclusively in operations. Since the situation in the coal-mining industry has been discussed at several recent meetings, that also will be passed over, and what follows should be taken as a discussion applying exclusively to the metal-mining industry.

Professional Development

In the operations field (both in mining and metallurgy) the work to which a beginner is assigned is seldom closely related to previous academic education. The contrast is almost as great as between

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[1] The Autobiography of John Hays Hammond, 1, 83–85. New York, 1935. Farrar and Rinehart.

studying ballistics in a classroom and then being given a golf club and a ball and being told to knock the ball into the cup. In mining it is only in the case of young engineers assigned to engineering work or sampling that the relationship of previous academic education to actual employment is at all close. The usual procedure in mining is first to put the young graduate at some simple underground task, such as cleaning track, trucking timber, etc., shift him about underground until he has become familiar with the workings and his muscles have hardened, and then put him at actual mining.

The professional development of a mining engineer to qualify him to assume operative responsibility involves three sequential stages: (1) academic instruction in fundamentals, (2) learning the details of practice, and (3) training in administrative work. Of these, No. 1 is the proper function of mining engineering schools; Nos. 2 and 3 are responsibilities of the industry, since the proper place to learn practice is in actual operating conditions, and training in administrative work can be given only where it is being done. The schools have to make reference to details of practice and methods of administration in their job of teaching fundamentals, but no one should suppose for a moment that this constitutes an adequate treatment of stages 2 and 3, which clearly are the responsibility of industry itself. Until this distinction of responsibility, and as to aim of procedures, is clearly recognized, misunderstanding will continue to exist as to what the industry has a right to expect of the schools, and what responsibility it should itself assume for training its operating administrative personnel.

One immediate aim of the work first assigned to a mining engineering graduate is to familiarize him with the details of his environment. The summer employment of an undergraduate in some mine does not meet this condition, and is only intended to make it possible for him to understand better classroom discussions in the school. Every teacher knows that a student returning from a summer's employment underground seldom has attained any comprehension of the functioning of the enterprise as a whole from his experience, and he is sometimes reported as an unsatisfactory employee because he did not know how to do things there was no rational basis for expecting him to know. Descriptive teaching in schools should be restricted to furnishing a background for instruction in fundamentals; it cannot be, and never should be expected to be, a substitute for that initial stage of employment which familiarizes the beginner with the details of procedure at the property where he is employed.

A more remote, but more important, aim of the tasks initially assigned is to qualify the young engineer for more responsible employment, which in mining usually means promotion to shift boss. It is necessary to raise the question here as to what are the fundamental requirements for the successful performance of the duties of a shift boss, and whether the tasks to which the young man is usually assigned operated to qualify him for them. Here the educational process in industry breaks down. Apparently it is fairly generally recognized that learning to be a good miner does not teach a man to be a shift boss, and that an excellent miner might be no good as a shift boss, yet the whole training process underground seems to be aimed at making a good miner out of a young engineer, and there is no provision, that I know of, for educating him to be a shift boss. I mean exactly what I say; that when he has become familiar with all the different kinds of work being done on his beat a young man with a natural talent for directing the work of others will be able to boss a shift, but there is no provision for teaching him to boss a shift.* Even in

* Foreman training courses, which are available in some industrial areas, but not in mining

mines where the foreman uses a roving assistant to whom temporary and special jobs of a supervisory nature may be assigned, such employment serves only as a check on whether the man has himself learned to be a shift boss, not to teach him to be one.

Gaining Practical Experience

A further drawback of the usual method of giving a young engineer "practical experience" is that it is not in practice conducted as a truly educational process. In education a beginner is given a task that is within his capacity to perform; as soon as he has mastered it a more difficult one is assigned (so that he does not lose interest through monotonous repetition), and so on until he has mastered the whole subject. The elements in this process, beginning with a simple task, are three: initial instruction and careful checking to ensure proper performance, prompt shifting to another task, and the use of a progressive sequence so that each conduces to the mastery of the succeeding one. Underground none of these are usually present; the task assigned is one that happens to be available, there is no checking beyond general observation by the shift boss, assignment to another task may wait on the young man's requesting it, and must in any case await an opening elsewhere. Thus it may easily happen that the fifth is simpler and easier than the first one.

It is generally assumed that a beginner is put under someone who will teach him what he is supposed to do, but in practice this works out imperfectly and mostly as a matter of chance. In theory the young engineer is put with a good miner who teaches him to be as good as himself; actually the good miner may have no capacity for teaching anybody else anything, and his disposition may be such that instead of trying to help the "college boy" he thinks it fun to ride him. The latter process has its educational values, but they do not conduce toward mastering the task in hand. What is interpreted as a "high hat" attitude on the part of the young engineer by his "buddy" may be due to shyness or embarrassment. A further drawback is that mining work is typically done on contract. One of the most important things for the young engineer to learn is the skillful operation of a rock drill and the proper placing of holes. His awkwardness at this in the beginning will slow up the work, so that the crew may not make any bonus above base pay, with the likely result that his "buddy" may perhaps keep him at shoveling, and similar tasks, and do all the drilling himself. In such ways a young engineer may spend considerable time underground without really acquiring any manual skill in some of the tasks he is supposed to be learning. In addition, it should not be expected that he will necessarily acquire skill at all of them, since the main objective is to learn enough about them so that, as a boss, he will know whether they are being done properly and how to correct faulty practice. This is an entirely different matter from being able to do them better than the workmen. It should be remembered that few singing teachers are especially good singers.

This is coupled with another problem. Much of the work in a mine is rather heavy and requires a strong physique to be able to do it. A boss needs brains, not muscles, but a beginner may be judged unfavorably simply because he is not strong enough for such work. I well remember in my own early experience being assigned to weigh and stack 350-lb. pigs of blister copper. My "buddy," whose education had not qualified him to add up the totals correctly, was fortunately about my own height and weight and was smart enough to have learned how to handle the

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districts, are usually either of the vocational type, intended for men of little academic education, or else discussions of general principles without those practical applications that are possible only in industry itself.

pigs adroitly instead of straining at them. We got on successfully together, but if I had happened to be teamed with a 200-pounder who handled them by main strength and awkwardness I might have been rated as a poor workman, and incidentally might have acquired a hernia. It seems probable that many engineers who might have made excellent mine superintendents have quit mining altogether because they found underground work too hard, and felt, for the reasons here set forth, that professional progress was too slow.

Progress is slow, not only because of the conditions already described but because some mine executives seem to entertain the belief that it is not practicable to have a boss who is not approximately as old as the men he bosses. That this is a fallacy is, I believe, demonstrated by Army experience, where properly trained young graduates of the Military Academy are commissioned lieutenants and give orders to sergeants, twice their age, who are as intelligent and as skilled in the performance of their duties as any miner. There is a little joking about "shave-tails," and discipline in the Army is different from that in industry, in that a man can be put in the guardhouse for disobeying orders. But I believe that any workman will willingly take orders from a young boss who really knows his business and has been trained to handle it. The trouble in the mining industry is not that young engineers are too young, but that the existing system does not function to qualify them to become shift bosses within a few years after graduation.

Qualifications of a Shift Boss

It may be that there exists an unconscious belief that shift-bossing is a task for which college graduates are not fitted. They have not been trained for it, but that they cannot be trained for it is also a fallacy. In this connection it is interesting to note that most of the recent appointees to the police force in New York City have been college graduates. The progressive mechanization of our mines and the use of more complicated and highly skilled procedures has long set a trend toward a need for abler and more effective supervisory control. In the face of this need young engineers have been too much relegated to technical staff activities, instead of building upon their academic training a capacity for administrative responsibility. The problem of providing for the promotion of men so trained is outside the scope of this discussion, though an important matter.

How does the management know when a young engineer becomes qualified to act as a shift boss? My guess is that mostly it has no positive criteria for judgment. One factor on which great stress usually is laid is "ability to get on with men," without any precise definition of what that means. It is evident that a young man might be hail-fellow well-met with every man in the mine and yet not be a good shift boss or foreman. Some of the best liked and most successful bosses, *per contra*, never seem to have much to say to anybody, yet their competence and keen sense of justice enable them to do their job well. In baseball there is the "grandstand player" and, in the absence of any positive means of checking either progress or the precise measure of executive ability in men, I believe executives are too prone to pick men for promotion by their visibility rather than their actual capabilities. I have no simple remedy for this condition, but it can never be corrected except through recognizing it frankly, and seeking a rational solution.

Summarizing, I believe that the usual working experience underground qualifies a young engineer to become a good miner, but does not qualify him to be a good shift boss. One may become a good miner without progressing toward being a shift

boss; on the other hand, a young man who makes a poor showing as a miner may be excellent material for supervisory work. Positive procedures for training a young engineer for supervisory work underground need to be worked out, as do positive criteria for determining his rate of progress and actual attainment. In South Africa the underground training of engineers is handled separately from regular operations, but conditions there are not similar to those in the United States, and the handling of "native" labor is so different a matter from that which we have that South African experience along this line can hardly be of more than general interest in the solution of our problem.

Planning and Organization

I have no definite program to put forward; partly because of a belief that more general recognition of this problem must precede effective attempts to solve it, and partly because it is probable that it will be worked out along different lines in different places. One concrete suggestion which can be made is that it should be recognized that training men to maintain and increase future production is as essential to continued effectiveness as it is to keeping up the daily output. Unless the president and principal officials of a company have this viewpoint the underground staff will find it difficult to do anything effective about it. Another is that such an educational program cannot simply be injected into the performance of routine operations, but requires some degree of separate planning and organization. Finally, I would suggest to the metal-mining industry that it consider the possibility of adoption, possibly in modified form, of the old educational technique of the seminar. Organized and planned meetings of young technical employees for presentation and discussion of technical problems of their own work offers much promise for education in analyzing and presenting problems, obtaining more than one viewpoint on them, and of exchange of information. This will not only facilitate the development of young mining engineers, but make them better satisfied, through a consciousness that they are developing professionally.

DISCUSSION

Tell Ertl,* New York, N. Y.—As a young mining engineer, who has worked in eight different mines in five western states, I feel that Professor Read, though covering his subject very well, has understated the case.

Perhaps it should be mentioned here that I have been a shift boss and later held a position in which I assisted in choosing new shift bosses at an important western copper mine. It was advantageous to me as a miner and a shift boss that my fellow bosses and the men working for me did not generally know that I was a "college punk" and that I would never talk of the fact to those who guessed that I was.

Undoubtedly the executives of this day are not as insistent as they were in John Hays Hammond's day that the job-seeking college graduate should have learned no mining in college, so consequently most young men graduating from mining schools receive employment from these progressive executives. Obtaining the job, though, generally consists of a note or a telephone call from the executive to the mine foreman to make a place for this embryo mining engineer.

Since the foreman is often not a graduate mining engineer—or, if he is, he is probably disillusioned by the tough row he had to hoe to become a foreman—he places the young man on some uninstructive job, preferably at the end of a drift away from all operations so that the young man will not get hurt. The shift boss chosen to guide this man's first cautious footsteps up the ladder of success is probably the boss who has the best reputation for getting a day's work out of every man each day.

Many case histories could now be given, but the gist of them is that the young man is kept mucking, hoisting timber, pulling chutes, or at other jobs of this type in which he must work hard and steadily, and in which he will

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have little opportunity to learn underground operations.

But all hope is not dead. This tired, hard-muscled young laborer, who is rapidly forgetting all his "larning," knows that sooner or later there will be an opening in the engineering or sampling department. Then, after he has sampled, drafted or surveyed long enough, he may be made a shift boss, if the need for shift bosses is acute. By then the only attribute he has besides his technical training that will make him a good shift boss is his toughness, because his training for the job will be negligible, his salary will not be enough to be stimulating, and his good will to the company will, if anything, be negative. (Within the past month I have received two letters from graduate mining-engineer friends, both of whom have had underground experience. One assures me that in another 18 months he expects to get on the staff either as engineer or sampler of a large copper mine in Arizona. The other informs me that the chances of improving himself were so slim at the mine in which he worked that he accepted a position as safety engineer for an insurance company.)

In short, I agree perfectly with all that Professor Read has said, yet feel that he has been a bit gentle in his treatment of the subject.

J. F. WEST,* Raleigh, N. C.—As I am new in the field of education, I cannot presume to prescribe a plan of post-college education for mining engineers. But I have had experience with post-college jobs in mines, and in giving my own case history I can illustrate many of the points brought out by Professor Read in his paper.

The discouraging outlook for jobs in 1932 was one of the factors that caused me to remain at the University of Wisconsin until 1933, when I received my M. S. degree in Mining Engineering. On the advice of a mining executive, I studied electrical engineering at the University of Illinois in 1933–34 but after one semester of this I took a job under the C.W.A. as engineering draftsman at the Illinois State Geological Survey. The work consisted of calculations of coal reserves from maps, computations of coal analyses on various bases, plotting of coal analyses to three coordinates, and mapping of coal areas. I was thrown into contact with geologists and engineers in a professional atmosphere that could not help interesting me in the work.

As the end of the C.W.A. loomed, I received an offer of a position as optical-pyrometer reader at the Illinois Steel Company's South Works plant, which I hastened to accept. There we read and recorded the rolling temperatures of the steel in the blooming mills, structural and alloy mills. As optical-pyrometer readers we were shifted about to different mills, where we obtained a working knowledge of the making, shaping and heat-treatment of steel from the iron ore to the finished product. Further, we became familiar with the everyday problems of management and labor relations and gained an insight into the struggle between man and the machine in the age of steel. Although I was conscious of the fascination of hot steel, I felt that I would never be content in the steel mills and quit after four months employment to seek work in the mines.

In the summer of 1934, I hitch-hiked to Lead, S. D., and tried my luck at the Homestake. Mr. Alexander Ross, the mine superintendent, said that the company hesitated to hire young engineers because they did not stay long enough to make it worth the company's effort and expense. In my own mind I agreed with Mr. Ross, because I had no intention of staying very long. Nevertheless I spent two months in trying to get a job, using my spare time in prospecting on the claims of a young engineer employed by the company.

In October I proceeded to Salt Lake City, where I obtained work at the Mountain City Copper Co., through Mr. J. O. Elton. At first, I was employed as laborer on construction of the surface plant. Later I was transferred to a drift heading underground, where I got my first taste of mining. We drilled with a Jackhamer, mucked with an Eimco-Finlay loader, and trammed with a storage-battery motor. But where I fell down was in cleaning up off the rough with a round-point shovel. In fact, it took me about a year of working in several mines to get used to the idea of hard physical labor.

After about a month at Mountain City Copper, the superintendent informed me that he was going to drive the 90 miles to town and

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that it would be a good chance for me to get out before we were snowed in for the winter. Though I was a little surprised at an invitation to leave, on reflection afterward I could see that with only four miners on a shift doing development work, the company could hardly afford to keep a man with as much inertia on a shovel handle as myself. Today, I am very grateful to the men who tolerated me as a raw "school of mines punk," for that month during which I was introduced to practical mining.

My next job began 10 weeks later at the Murchie mine, a subsidiary of the Newmont Mining Co. at Nevada City, Calif. I was employed as a mucker and periodically moved around to different stopes, raises and drifts, so that in the 10 months of my stay I became quite familiar with the mine. My shift boss was Louis Saban, a Slav who speaks with a foreign accent but is probably the best mine boss I ever met. By this time I realized how easy it was to lose a job and actually learned to muck, timber, drill and blast, but I grew impatient of my slow progress and quit in November of 1935.

Considering that if I were to remain a mucker I might as well muck in a variety of mines, I journeyed to Virginia City, Nev., to see the famous Comstock. I was not long in obtaining employment as an extra man in the mill of the Sierra Nevada Consolidated, but owing to financial problems the company suspended operations two weeks later. Had I been interested in milling as a career I might have followed the leads given me at this mill, but mining attracted me more and I was soon employed as mucker at the Dayton Consolidated mine at Silver City, Nevada.

My work at the Dayton was much the same as at the Murchie, except that in some stopes the Dayton used square sets. Because the force was reduced following changes in production rates at the several properties, I was laid off with five other men in February and returned to Nevada City.

I had absorbed something of the spirit of the 10-day miner, who is rapidly becoming extinct, and felt the urge to work in ever new quartz veins. In the spring and summer of 1936 I helped reopen the Sneath and Clay mine, closed for about 30 years, trammed at the Lava Cap mine and mucked at the Original Sixteen to One mine in Alleghany, Calif. During that period, in addition to becoming familiar with mining problems new to me, I was able to observe the intricate relations of the men where high-grade ore is concerned.

An offer to help my major professor at the University of Wisconsin, R. S. McCaffery, on mine examinations in Alaska brought me back to the more technical aspects of mining engineering. This was, indeed, one of the pleasantest as well as most educational aspects of my life, since two of us younger men were using all the resources at our command under the guidance of an able engineer.

With the examinations finished, I was thrown back on the labor market and found employment as a mucker at the Sunshine mine, near Kellogg, Idaho. I learned a good deal at that property and have never ceased to marvel and speculate on the remarkable ore body. But by that time I was looking for a place to settle down and the company seemed to be ably staffed by young men. The days of expansion in the mineral industry, I realized, were over, yet my chief hope for advancement seemed to be in locating with some company where operations were just starting.

I obtained a job as miner in reopening the Tamarack and Custer Consolidated mine near Wallace, Idaho, and remained with the company until September 1937. My "pardner" happened to be an Italian whom I had known in California and our associations were very pleasant. Our work took us throughout the entire mine, retimbering shafts, raises, and caved drifts as well as following the new headings. The management seemed congenial and understanding, but the history of the lead-zinc district was one of feverish activity alternating with periodic curtailment of production and complete shutdowns.

The desire to make a home began to play a growing part in my plans and with the dependence of lead-mining activity upon fluctuating prices I could have little faith in that field as a career. Consequently I lost interest in my work, quit, and drove to Arizona.

My intention had been to seek employment in the copper mine at Jerome, Ariz., but upon my arrival I learned that the price of copper had dropped four cents since I had left Idaho and that the United Verde had laid off 200 men. There remained little choice for me and I drove

to the gold districts, where I obtained a job as sampler at the Tom Reed gold mines at Oatman.

The work at Oatman was very interesting for a while, but there seemed to be no future that a man could count on in the camp. The talk of going off the gold standard as well as of raising the price of gold, together with the ultimate exhaustion of every medium-sized ore body, made any long-term outlook extremely uncertain.

In the midst of these perplexities I quit my job and went prospecting. Prospecting fascinated me and afforded the most healthful living conditions I can imagine. But it did not make me any money.

After six months on my own I had to get a job again, and this time was fortunate in making connections at Park City Consolidated, Park City, Utah. I began as a mucker in a contract drift with the prospects of engineering work later. In the two months at the mine I witnessed some very interesting engineering work, some applications of geology and a satisfactory handling of a difficult labor situation. Finally Mr. Gloyd M. Wiles sent me to central Idaho as engineer with a placer-testing party. This was indeed the type of work I had dreamed about some years before, and was an opportunity to work up. We did the work creditably enough but there did not happen to be enough gold in the gravel and the option was relinquished. I was offered employment at Park City again, but I had had enough. I could not see the establishment of a home and satisfactory family life there or in the majority of mining camps of my acquaintance, and the fortunes of mining were too fickle to hold my interest by themselves.

I took one last fling at prospecting in Arizona during the winter of 1938–39, at which time I decided that the odds are too long against the prospector. I came to the conclusion that my life as a tramp miner had been both thrilling and educational, but that my most urgent need was to take root somewhere and become a permanent part of my environment.

I am now an instructor in geological engineering, have begun to make a home, and believe that I have found a field that is making a most valuable contribution to our country's future. However, I would be happier if I could predict for my students a less precarious existence than part of mine has been.

Settling Device for Sludge Samples

By A. A. Gustafson,* Member A.I.M.E.

(New York Meeting, February 1943)

In diamond or churn drilling for the prospecting of ore bodies, two products can be used for quantitative analyses; i.e., the core and the cuttings, or sludge. Some operators prefer an assay of core; others insist that the sludge be sent for analysis. Both must recover the sludge for check samples or for assays if there is no core recovery for any particular section of the diamond-drill hole.

The Freeport Sulphur Co. found that numerous cumbersome sludge boxes or barrels were inconvenient and delayed drilling. The sludge settled slowly in the ordinary shallow, rectangular diamond-drill boxes and its total removal from the container was difficult, time-consuming and unsatisfactory; and sampling of deposits of which the metal contents are easily concentrated, as in ores containing cinnabar, gold and silver, is subject to error if the boxes are not thoroughly cleaned.

Mr. Charles Ballard, dredge superintendent at Freeport's operation at Grande Ecaille, Louisiana, developed a portable cone while he was in charge of prospecting a mercury property near Tonopah, Nevada, in 1940. Ballard later improved the design while the settler was being used at a manganese property in New Mexico.

The operation is based on the "free settling" of comminuted particles in a quiescent liquid. The ordinary sludge box discharges over a lip 8 to 12 in. wide and consequently the water has sufficient velocity so that the discharge water is never clear. The cone settler used by Freeport Sulphur Co. discharges over the entire circumference, which gives a discharge lip of 9 ft. in the size of cone used at the cinnabar and manganese prospects. Consequently, the velocity of the discharge water is slow enough so that all fine material settles and the discharge water is clear. When the drilling of a sample interval is complete, the sludge can be drawn off by the time the driller is ready to cut another sample. Thus, only one cone is needed per drill.

The cone has been used by Freeport successfully in churn-drilling a manganese deposit, in diamond-drilling a mercury deposit, and in diamond-drilling a deposit of ore containing tin, copper, gold and silver. The cone is greatly superior to the old methods of collecting sludge and is now used on all Freeport drilling campaigns in the western states, both for surface and underground drilling. Fig. 1 shows the important details of a cone designed for average operating conditions, 30 in. in diameter and 36 in. high. Abnormal conditions may require some revisions in design. Fig. 2 shows a working drawing.

A hose or pipe larger than the one used to convey water from the pump to the swivel head of the string of rods is connected to the stuffing box at the collar of a diamond-drill hole. The usual T-connection is used. It is not necessary to use a stuffing box if the top of the cone can be placed lower than the collar of the hole. Then the return water and cuttings can be made to flow by gravity. This hose or pipe discharges the cuttings of the drill bit and the return

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* Freeport Sulphur Co., Port Sulphur, La.

water into a small cone (1 on Fig. 1). At the neck this cone at 8 extends at least 12 in. below the overflow water level at 3. This prevents surging within the cone, confines the sludge particles to the center portion 13 of the main cone 9.

Clear water overflows from passage 6 over lip 3 into a launder 4. The shallow cylinder 2 prevents overflow of any fine floating particles, which can be skimmed from the surface and added to the remainder of the collected sludge. The overflow pipe 5 should have male hose connections, brazed or soldered, in order that the sections of ordinary garden hose that convey the water to the sump or supply tank may be quickly removed during moving operations.

The rake teeth 11 are made of 20-gauge galvanized sheet iron, circular in shape at the lower scraping edges, with angles on the upper sections for riveting to a flat bar 10. The rake teeth should be placed on the bars at an angle of 45° slightly lapping each other. Two such assemblies are required, the teeth on each rake being offset in relation to the teeth on the adjacent rake. When these assemblies are placed in the cone 9 and connected to the hinge joints 15, which are supported by spider 16 and screw shaft 17, they form inclined rakes toward the center of the cone at 13. When operated by the handle 12 the rakes move all of the sludge deposited on the bottom of the cone to the center discharge hole 13. These rakes are used only at the time of removing the sludge, to ensure complete removal of all cuttings and to prevent the building up of accretions on the bottom of the cone.

It is necessary that the discharge valve 14 be a gate type, or one of similar operation, in order that the settled sludge may flow freely without obstruction. The $\frac{1}{8}$ by $1\frac{1}{2}$-in. flat bar 25 with $\frac{5}{8}$-in. round shank threaded at upper end and screwed and locked into the lower end of spider 16 will prevent clogging through hole 13 when sludge is being drawn off. The settled sludge may be drawn off in ordinary 12 or 14-qt. buckets. During recent operations underground, Freeport found that in tak-

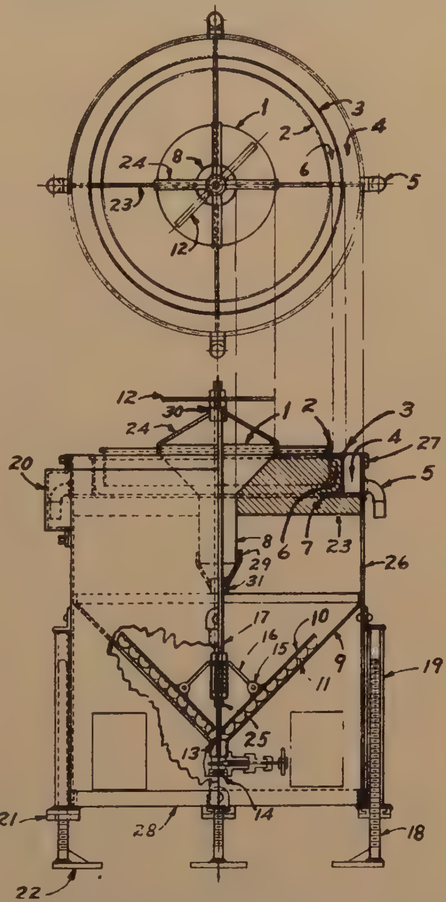

Fig. 1.—Details of sludge-settling device.

ing 20-ft. samples from a hole using an EX bit, the usual number of buckets was seldom less than four or more than six. In taking the samples from the settling cone, the first one or two bucketfuls is naturally quite thick, but as soon as the last of the cuttings is discharged from the cone the water flows clear. This, of course, enables the most inexperienced operator or helper to do efficient work.

The threaded 1 by 20-in. legs 18, which are fitted into 1-in. galvanized pipes 19 sup-

port the cone and are for the purpose of levelling the cone on uneven ground, so that there is an even discharge of clear water over the entire circumference of the

The settler is light enough in weight so that two men can easily carry it to any place in the mine or on the surface. For shipment, foot pieces 22 can be turned

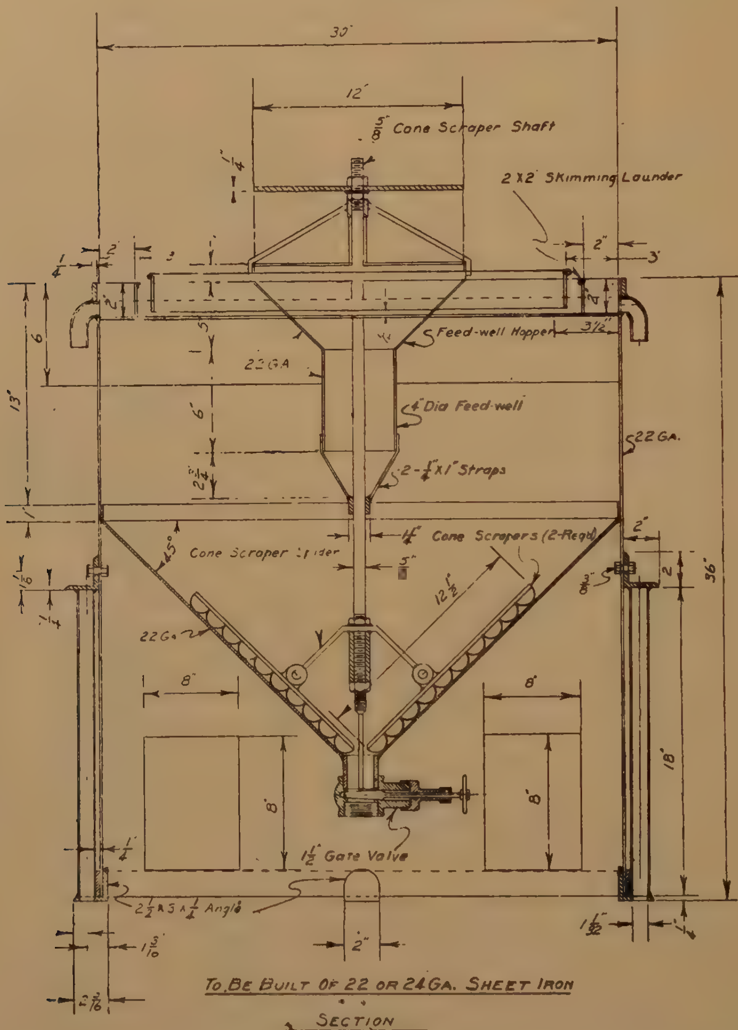

FIG. 2.—THIRTY-INCH GALVANIZED SHEET-IRON CONE PIT FOR SETTLING TANK.

cone. It is only necessary to have the cone high enough to ensure easy passage of the buckets. After the buckets have been allowed to settle for an hour or more, the excess clear water can be siphoned off with a small rubber hose.

inward and locked by nuts 21. With covers 20 placed over discharge pipes 5, the settler can be rolled about like an oil drum. There is nothing complicated about the device; it is constructed of materials easily obtained and may be fabricated by any first class

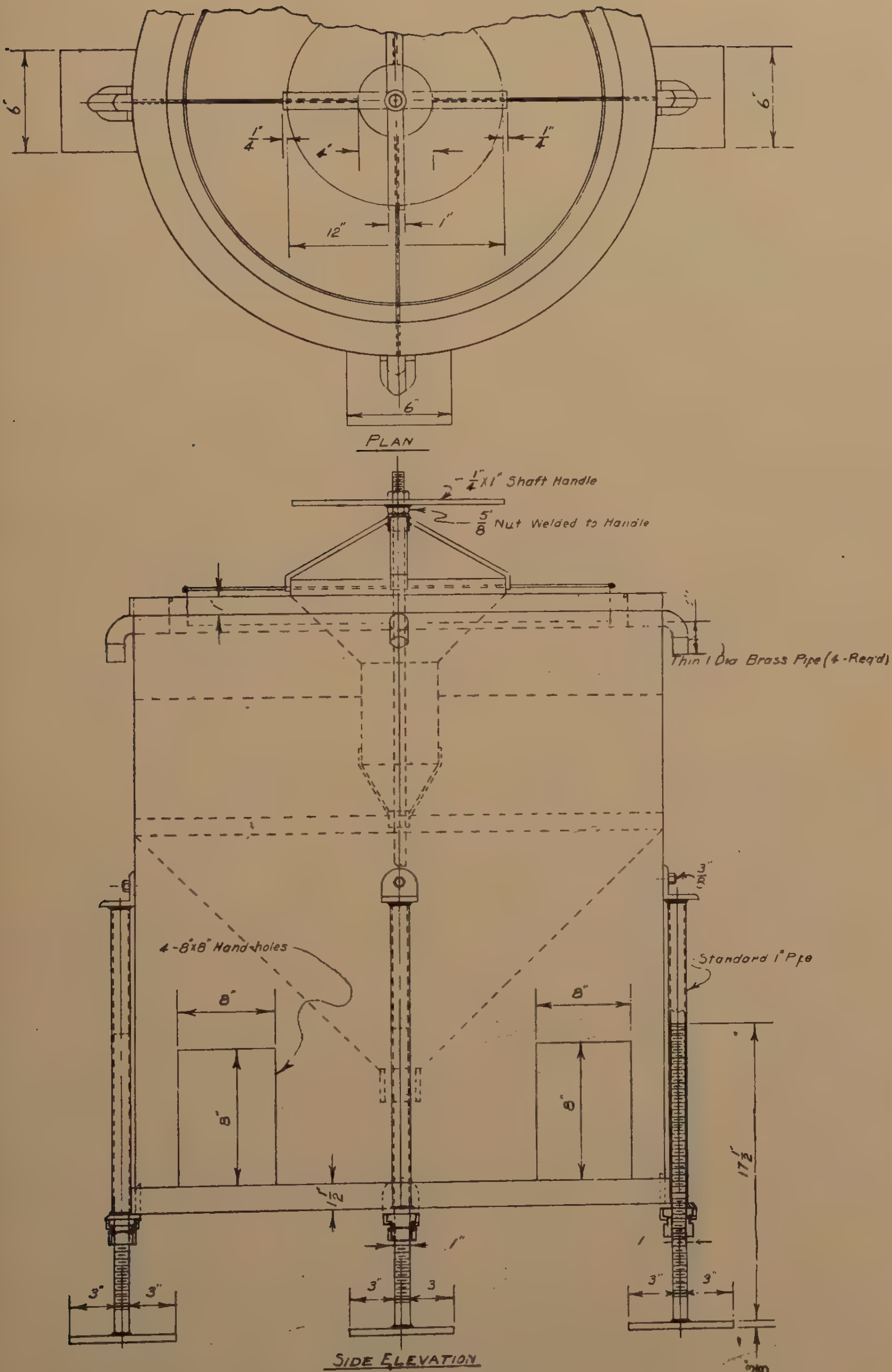

FIG. 2.—(*Continued*)

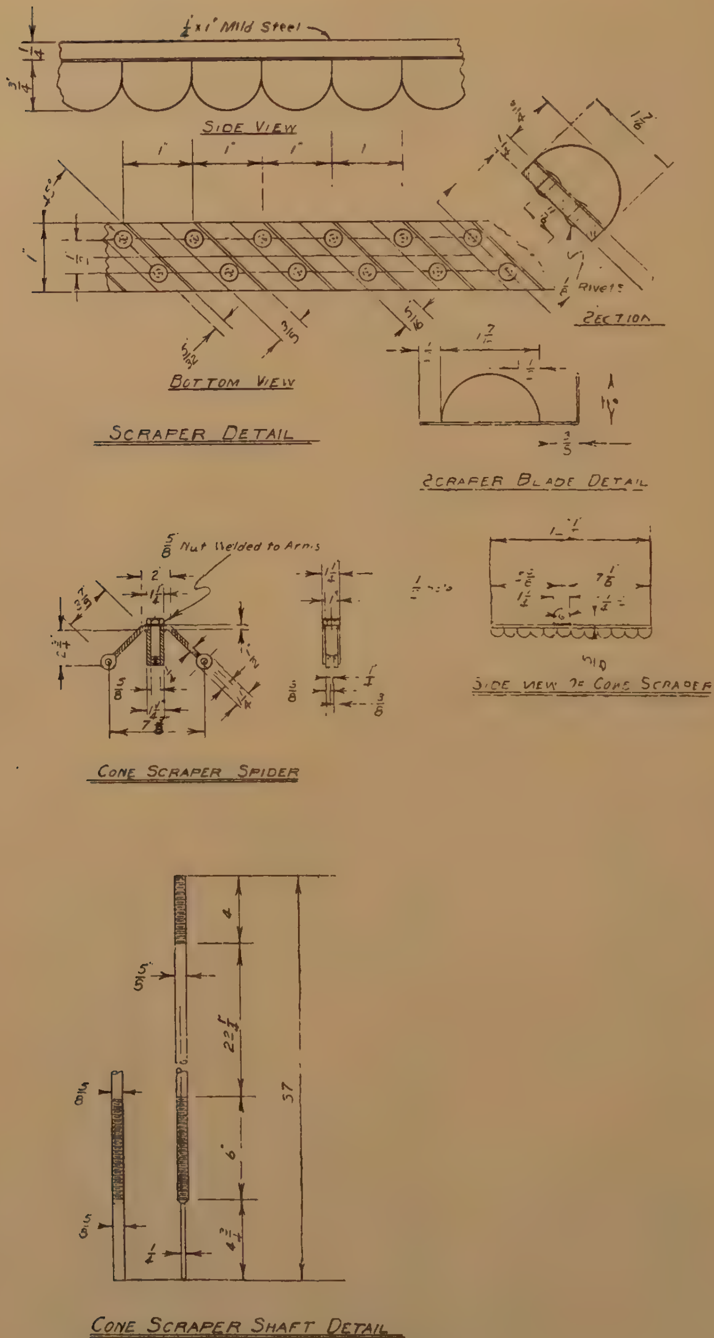

FIG. 2.—(*Continued*)

mechanic. The entire main cone body 26, cone 9, small cones 1 and 8, skimmer 2, launders 7, 3 and 4, and spreaders 23, should be of 20-gauge galvanized sheet iron. The 3/16 by 1½-in. iron bands riveted around the main body of the cone 26 at 27 and 28 give additional strength. Other fixtures (12, 24, 29 and 16) may be ¼ by 1-in. iron. The shaft 17 is a ⅝-in. rod threaded and locked into handle 12 and spider 16, guided through two bearings 30 and 31.

Foremen of Boyles Brothers Drilling Co., of Salt Lake City, who have done the drilling several times for the Freeport Sulphur Co. on prospecting work, report that this is the most convenient settler they have ever encountered.

Anaconda Copper Mining Co. has borrowed one of the settlers from Freeport to try it out on drilling at Copper Canyon, Nevada. Some Bureau of Mines engineers have also requested information about the settler.

DISCUSSION

(*T. B. Counselman presiding*)

R. D. Longyear,* Minneapolis, Minn.—In my opinion, when drilling under sampling procedure in which the total weight of sample must be ascertained, this device has an advantage over devices that cut out a fraction of the total sample.

J. A. Younkins, Jr.,† Pittsburgh, Pa.—Possibly the settling rate of solids in this device might be improved by the use of causticized starch.

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* President, E. J. Longyear Company.
† Duquesne Light Company.

Determination and Localization of Metallic Minerals by the Contact Print Method

By Gregoire Gutzeit,* Member A.I.M.E.

(New York Meeting, February 1942)

The development reported in this paper was begun by the author a number of years ago, while he was a lecturer on complex chemistry and metallurgy at the University of Geneva, Switzerland, and in charge of that institution's research laboratory. Completion of the development and reduction to practice did not occur until last year at the Westport mill, laboratories and testing plant of the Dorr Company, where this method is now in general use for identifying and localizing the mineral constituents of both metallic and nonmetallic ores. It has proved to be a practical tool of the metallurgical staff at the Westport mill, devising unusual milling flowsheets for complex and refractory ore bodies. In a number of instances the findings have been successfully used as fundamental factors in the design and specification of complete metallurgical plants. Seven examples of the use of this method in solving practical milling problems are included in this paper.

General Outline

The microscopic study of an ore, at first only of scientific interest, has proved extremely useful both in practical geology and in ore dressing. The binocular magnifying lenses for rough observation of the products of an ore-dressing plant have been in use for a long time; but only within the last few years has a systematic microscopical examination of ore samples become a normal tool of any research laboratory dealing with ore-dressing problems. Such an investigation avoids considerable loss of time as well as empirical trials and useless experimentation. It will generally indicate the correct grinding and often will suggest the proper method of treatment.

The petrographic determination of transparent minerals in thin sections, or even in grains, has been perfected, and very accurate methods have been devised to meet any requirement, such as Feodorof's rotating stage on the petrographic microscope. Almost all transparent minerals can be positively identified by their optical properties; i.e., isotropism, anisotropism, color, pleochroism, refractory index, cleavage, crystal form, birefringence, and others.

The identification of opaque minerals is a much more difficult problem and can be done only by using polished specimens. Unfortunately, most of the valuable metallic ores, and even some of the nonmetallic ones, are not transparent; therefore it has been necessary to develop a special technique for the determination of these minerals. Some optical properties—for instance, isotropism, anisotropism, color, cleavage, reflecting power—are often adequate in roughly classifying the mineral. Only "reflecting power" is a measurable value, provided special photometric equipment (ocular photometer or Orcel's photoelectric cell) is available. The other optical properties mentioned are more or less relative,

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therefore comparison plays a greater role than the expression of quantitative figures. It frequently happens that several interpretations are possible, as the mineral to be identified often has a certain number of characteristics closely related or similar to those of other minerals,[1] so it is necessary to revert to chemical tests. Two different procedures, somewhat complementary, have been used: etch tests and microcrystalline reactions.

The etch method, which shows the sensitivity or insensitivity of the mineral toward certain reagents, can further limit the classification but the results of these tests are often dubious and, indeed, different specimens of the same mineral frequently give different etch tests. Furthermore, if the drop of reagent covers two different mineral particles, a so-called "concentration cell" may be formed, which, by means of an electrical phenomenon, will produce a corrosion that is not predicted by etching of these mineral grains when separately tested.

Microcrystalline reaction depends upon the formation of characteristic crystalline precipitates that positively identify an element. This method has the advantage over the etch tests in that it gives a complete picture of the elements present in the mineral, provided the latter can be scratched out and the resultant powder dissolved in an acid.

These procedures are rather slow and are reliable only in the hands of an expert, but their greatest disadvantage is that the polished surface is destroyed with each test, thus making any further microscopic examination impossible without regrinding and repolishing. The new face being different from the first one, no localization of the elements is possible.[2]

In order to develop a process that would be sensitive, and at the same time quick and simple, the print method, both contact and electrographic, has been adapted to the mineralogical field.[3] This method chiefly uses specific organic reagents, giving colored complexes with one or a few elements, and can be summarized as follows:

A piece of gelatin-coated paper is impregnated with a selective attacking reagent. It is then placed on top of the polished surface and pressed down in contact with the specimen. After the paper is removed it is developed in a reagent that is specific for the questionable ion. An inverted image of the position of the element on the polished surface thus results.

The advantages of this print method are apparent. In the first place, a true picture of the localization of each element on the surface is obtained, which helps considerably in determining the state of aggregation of the ore. Also, it shows the presence or absence of a given element in each mineral particle by one test only, while by etch tests or microcrystalline tests each mineral grain must be tested separately. Furthermore, the polished surface is not destroyed. Usually about ten prints can be made with one polish, and it is easy to restore the original smoothness merely by rubbing the specimen on the felt. For a person skilled in this technique, a clear conclusion is more easily and more definitely reached by the print method than by any other procedure. In addition to these direct advantages, records can be kept for every sample, as the prints, when thoroughly washed, will last indefinitely. Furthermore, photomicrographic enlargements of both the sample and the print will show (especially when the proper filters are used) many details that escape observation with the naked eye, and permit the superimposing of the images of the sample surface and the print, thus giving a perfect register.

This general outline shows that the method is a practical tool, and actually will solve many problems in geology and ore dressing. The procedure makes it possible to locate important secondary constituents (for example, silver or nickel) in an ore and to show their continuous association with a

[1] References are at the end of the paper.

certain type of mineral. The extent of interlocking of two different minerals that otherwise could not be distinguished from each other can also be studied. It shows whether the losses in concentration tailings are due to the presence of oxide minerals, to insufficient liberation of very similar species, or to the presence of finely disseminated extraneous particles in the mineral to be concentrated. This method is often the key for outlining a process of differential flotation. The same information cannot be obtained by ordinary technique (microscopic observation and chemical analysis) as it does not show the localization of the different elements present in the ore.

Historical Summary

A print method was used by Heyn[5] and by Baumann[6] as early as 1906 in the metallographic field, to detect sulphurous segregations in iron and steel. The procedure was perfected recently by Niessner,[7] who utilized specific reagents for the study of alloys. A. Glazunov[8,9] developed the electrographic attack of metallurgical products, using cardboard to block the cations migrating from the anode to the cathode. The contact print method, with chemical attack and specific organic reagents, was first adapted for the determination of opaque minerals by the author and his co-workers, M. Gysin and R. Galopin,[3] and carried on by him with R. Galopin,[10] and by the latter alone.[11] P. Wenger, G. Gutzeit and Th. Hiller[4] applied the electrographic attack, initiated by Glazunov, to insoluble minerals. Previous to that study, R. Jirkovsky[13] had already proposed the electrolytic attack on minerals, but the identifying reagents used by him were not specific. Th. Hiller[14–16] (former engineer of the Dorr Company and G. Gutzeit's assistant in the Dorr Laboratory at Geneva, Switzerland) reports, in his doctor's thesis, the very complete and excellent work he did under the direction of M. Gysin and the author.

The specific reagents used in the print method have been developed for the so-called "spot-test procedure" by numerous workers since 1922. Among the publications covering this field, the articles and books of F. Feigl[17] and G. Gutzeit[18,19] can be mentioned. I. Mellan's book[20] may be recommended among the recent American literature. Some condensed information is also available in the last edition of Lange's "Handbook of Chemistry."

Principle of the Contact Print Method

As already outlined, gelatin-coated paper (i.e., Eastman Kodak Co. paper, "Kind 867," or a glossy photographic paper from which the silver salts have been dissolved by thiosulphate and thoroughly washed out) is impregnated with an attacking reagent, the excess of which is *removed* by blotting paper. It is then laid face down on the polished surface being studied. Evenly distributed pressure is applied on the paper, for a period of time varying with the reagent used and the minerals present in the sample. If the electrographic process is advisable, the time of attack generally can be greatly reduced, because of the phenomenon of anodic dissolution.

Thus, a very thin superficial layer of the polished surface is dissolved and the respective ions are blocked in the gelatin layer of the paper, which acts as a colloidal diaphragm. The surface layer of a polished mineral is much more soluble than the natural mineral itself, probably because of a mechanical loosening in the crystal lattice.[1] This is confirmed by the fact that certain minerals cannot be etched by a given acid, but this same acid will dissolve enough of the mineral to give a print.

After the attack, the paper is developed in a specific reagent solution, which gives a colored spot corresponding to the position of the element in question, if the latter is present in the mineral. The print is an exact but inverted image of the polished

section. In some cases it is possible to soak the gelatin-coated paper with both the attacking solution and the specific reagent, thus arriving at a finished print in one operation. This type of image is always clear and never diffused, thus being extremely suitable for microscopic and photomicrographic work.

It is evident that such a process can be possible only when the reagents used to identify a given element are extremely sensitive and specific. These two conditions are usually fulfilled by organic compounds suitable for forming inner complexes (chelate rings). The following examples will illustrate this fact:

0.0006 mg. of chromium can be detected by Alizarin R. C.
0.00005 mg. of cobalt can be detected by Alpha-nitroso-beta-naphthol
0.00003 mg. of cobalt can be detected by Nitroso-R Salt
0.0001 mg. of copper can be detected by Alpha-Benzoinoxime
0.00005 mg. of copper can be detected by Dithiooxamide
0.00004 mg. of lead can be detected by Diphenyldithiocarbazone
0.0002 mg. of manganese can be detected by Benzidine
0.000001 mg. of manganese can be detected by Tetramethyldiaminodiphenylmethane

The preceding description shows that the procedure for making contact prints usually comprises two consecutive operations: (1) the attack of the mineral, resulting in the dissolution of its elements; (2) the identification and localization of the constituents by means of specific reagents.

Attack of the Mineral

The technique suitable for metallographic work cannot be used on polished ore samples, as generally they are very irregular in form; and because of this fact, the pressure cannot be evenly distributed when they are pressed face down on the cardboard. To get the best results, the moist gelatin paper is placed on the surface of the specimen, which is embedded either in plasticine or in a molded plastic with planed faces, and put in a press with parallel plates. The upper plate is weighed down with one kilogram or more, depending on the smoothness of the surface. Some elastic medium must be inserted between the paper and the lower plate of the press, preferably a piece of felt and/or pliable rubber. On very soft minerals surrounded by hard ones, the surface will never be smooth. For those, the best results are obtained by placing a flattened bit of plasticine or kneaded eraser wrapped in tinfoil between the paper and the press.

As stated, time of attack varies with different minerals as well as with the dissolving reagent. In a general way, it ranges from 2 to 5 min., but may be as long as 10 min. for very refractory compounds. The longer the attack, the more diffused will be the print; so it is always better to use strongly concentrated attacking reagents, provided the latter are selective and will not destroy the gelatin layer of the paper.

In order to overcome this defect resulting from prolonged attack, and also for very insoluble minerals, electrolytic dissolution is preferable. However, this requires a much more elaborate setup and, of course, can be used only on minerals that are good conductors of electricity. The principle of the electrographic process is as follows:

The mineral is connected with the positive pole of a battery; the gelatin paper, soaked with an attacking reagent (an electrolyte), is pressed down on the polished surface by means of a conducting medium (thin metallic foil) which is connected to the negative pole. The mineral is attacked by anodic dissolution. The electronegative elements are ionized and, when migrating between the mineral (playing the role of an anode) and the metallic foil (cathode), are absorbed in the gelatin layer. The electrolytic dissolution is much stronger than the chemical one, and, as a result, the surface is

sometimes slightly corroded when the attack is too long. For this reason, it is very important to reduce the time of contact to a minimum.

If the ore sample is a compact piece of any conducting mineral, the best procedure is to use a brass cup somewhat larger than the specimen, connected to the positive pole of the battery. The sample is pressed into a ball made of crumpled tinfoil and placed in the brass receptacle. For a more permanent arrangement, the mineral can be embedded in an alloy that has a very low melting point; the Lipowitz alloy,* liquefying at 70°C., is well adapted. The gelatin paper is pressed down on the polished surface of the sample by means of a piece of soft rubber wrapped in a very thin aluminum foil connected with the negative pole of the battery. It is advisable to place a piece of porous rubber under the sample in order to get an even pressure.

If the specimen is not compact, but contains grains or veins of conducting minerals, the contact can be established by a fine needle connected with the positive pole of the battery and touching the mineral at one point.

The intensity of the electric current must be less than 50 milliamperes. Generally a potential from 2 to 6 volts is sufficient but in extreme cases even 16 volts may be necessary. If a storage battery is used, a variable resistance of 30 ohms, and a galvanometer, must be placed in the circuit.[14–16]

The Reagents

As stated, two separate reagents are generally required in order to obtain prints—the selective attacking reagents and the specific reagents for the detection of each ion.

Attacking Reagents.—The choice of the reagent depends essentially on the nature of the mineral, and is a matter of knowledge and experience. Often it must be established by preliminary tests. Hydrochloric and nitric acids, alone or combined, sulphurous acid, organic acids, ammonia, cyanide, and caustics are used most. An oxidizing agent, such as bromine or hydrogen peroxide, sometimes is required. In order to get very decisive results, the attacking reagent should be selective as far as possible. This may be illustrated by the following example:

Assuming that an ore contains copper, cobalt, antimony and silver, the latter element being an accessory constituent, ammonia will attack the copper and the cobalt, forming complex ammines, and both can be revealed by the same specific reagent (dithio-oxamide). Antimony will be oxidized to antimoniate by caustic potash containing some hydrogen peroxide. A cyanide solution will dissolve the silver. The papers are then developed in the proper reagents, and the localization of each element is shown on a corresponding print. By finishing photomicrographs of both the sample and the different prints on transparent paper, and superimposing them, associations of the different elements can be shown.

Specific Reagents.—The specific reagents are chiefly organic products forming inner complexes, and have been developed for the qualitative "spot-test procedure." The essential conditions for their use in the print method are the following:

1. The resulting combination between the element in question and the reagent must be colored and, if possible, insoluble.

2. The reagent must be extremely sensitive, as the amount to be tested is very small. This point has already been discussed.

3. The reagent must combine only with the desired element and not with others.

The last condition is the most difficult to arrive at, as actually there are only a few known chemicals that react with only one

* Lipowitz alloy contains 44 per cent bismuth, 23.5 per cent lead, 23.5 per cent tin and 9 per cent cadmium.

element. Nevertheless, it is possible to increase the selectivity of a reagent by masking the objectionable ions. This can be done (*a*) by using a supplementary reagent that forms a soluble and stable complex with the objectionable ions; (*b*) by using an additional reagent that will oxidize or reduce the objectionable ion, provided that its highest or lowest valence form will not react with the specific reagent; (*c*) by using a secondary reagent that will form an insoluble precipitate with the objectionable ion, thus preventing its combination with the specific reagent. For example, in order to mask iron, phosphoric acid can be added to the attacking reagent. Iron will be fixed as $FePO_4$, which will not react. Tin can be complexed by hydrofluoric acid, all the metals of the iron group by organic oxyacids, cobalt by cyanides, mercury by chlorides.

Determination of Principal Elements in Opaque Metallic Minerals

As explained, the choice of the attacking reagent must be adapted to the composition of the mineral. Thus, berthierite ($FeS \cdot Sb_2S_3$) can be dissolved by a 40 per cent solution of caustic potash; mispickel ($FeAsS$), by nitric acid and hydrochloric acid (20 to 35 per cent); smaltite-chloantite ($CoAs_2$-$NiAs_2$), by concentrated ammonia. It is almost impossible to indicate the attacking reagent best suited for every problem. The examples in Table 1, comprising some antimony-bearing minerals, show the complexity of the question.

TABLE 1.—*Attacking Reagents for Some Antimony-bearing Minerals*
ELEMENT SOUGHT IS ANTIMONY

Mineral	Attacking Reagent	Electric Current, Volts	Time, Min.
Stibnite	Tartaric acid and nitric acid		3
Tetrahedrite (Fahlerz)	10 per cent tartaric acid and phosphoric acid	8 to 12	1
Ullmanite	Tartaric acid and nitric acid	8	1
Ullmanite	20 per cent nitric acid		5
Berthierite	40 per cent caustic potash		3
Boulangerite	40 per cent nitric acid		3
Bournonite	Hydrochloric acid and nitric acid, diluted 1:1:2		3

A limited choice of specific reagents for the most important elements is given below:[14–20]

Antimony.—9-Methyl, 2, 3, 7-Trioxy-6-fluorone, in a weak acid solution, gives a red precipitate. Iron also reacts, but it can be masked easily by the addition of phosphoric acid. (In neutral solutions, other ions interfere.) It is advisable to add some tartaric acid, as well as the phosphoric acid, directly to the attacking reagent.[21–24]

Arsenic.—N-ethyl-8-hydroxytetrahydroquinoline hydrochloride gives a reddish brown complex with arsenious ions.[25] Copper, lead and mercury interfere, but can be masked by the use of an excess of chloride. It is also possible to detect arsenic by oxidizing it to arsenate, using an alkaline attacking reagent (caustic soda or ammonia) to which hydrogen peroxide has been added. It is then developed in a silver nitrate solution (1 to 2 per cent), thus giving the brown silver arsenate.[26] Furthermore, it is possible to reduce the arsenic ions to metallic state by stannous chloride (Bettendorf's reaction).

Bismuth.—In acid solution, bismuth gives, with cinchonine (alkaloid) and potassium iodide, an insoluble double iodide, red-orange in color.[27–29] Copper interferes by liberation of iodin. If copper is present, after the reaction the gelatin paper must be washed in a 10N solution of thiosulphate.

Dimethylglyoxime, in ammoniacal solution, gives a yellow print.[30] This reaction is disturbed by the presence of nickel, which can be masked by cyanide.

Cadmium.—Dinitrodiphenylcarbazide gives a brown to blue-green precipitate with cadmium, in an alkaline solution and in the presence of potassium cyanide and formaldehyde.[31]

Paranitrobenzenediazoaminoazobenzene,[32] in the presence of potassium and sodium tartrate, gives, in acetic solution, a pink-violet print with cadmium-bearing minerals.

Chromium.—Acid alizarin RC gives an orange reaction.[33] After developing, the paper must be washed in 2N sulphuric acid and

neutralized with ammonia gas (by holding the print above an open ammonia bottle).

If the attack is made with an oxidizing alkaline reagent (for example, caustic soda and hydrogen peroxide), the chromium is transformed into chromate. For this, the following reagents are recommended: (1) silver nitrate gives a red print; (2) alpha-naphthylamine (in hydrochloric solution) gives a violet-brown color.[34]

Cobalt.—In a weak acid solution alpha-nitroso-beta-naphtol yields an insoluble brown complex.[35] Iron also reacts, but if the attack is performed with ammonia this interference will be suppressed. Nevertheless, if the mineral is insoluble in the latter reagent, the iron can be masked by phosphoric acid.

Dithio-oxamide (rubeanic acid) gives a tan print in weak acetic or ammoniacal solution.[36] Nickel and copper also react; but a 1 per cent cyanide solution dissolves the inner complexes of these ions, leaving a clear cobalt print.

If an electrolytic attack is used, potassium cyanide alone gives a direct yellow-orange print of cobalticyanide of cobalt.[37] This reaction is not very sensitive, but extremely specific, and thus may be recommended for the determination of cobalt in cobalt minerals, such as smaltite and safflorite.

A very sensitive reagent for cobalt is Nitroso-R-Salt[38] in a solution buffered by sodium acetate, giving a red print. Copper, iron, nickel and other metals also give reactions with this compound, but by a very short wash in 2 : 1 nitric acid these are destroyed, while the print of the cobalt remains unchanged. Of course, the paper must be washed in water after the acid treatment in order to prevent it from turning yellow.

Copper.—In neutral, ammoniacal and acetic solution (even of high concentration), copper forms a dark green insoluble inner complex with dithio-oxamide (rubeanic acid).[36] The reaction is extremely sensitive. The intensity of the color is proportional to the copper content; metallic copper giving an almost black print. A large excess of cobalt is disturbing, and under this condition another reagent must be used.

Alpha-Benzoinoxime, in ammoniacal solution, gives a green complex with copper.[39] If the attacking reagent is ammonia, the reaction is specific. Otherwise, sodium potassium tartrate must be added in order to mask the metals of the iron group.

In acetic solution, salicylaldoxime yields a green print.[40] Only gold and palladium can interfere.

It may be pointed out that, with copper, potassium ferrocyanide, in neutral or acetic solution, gives a brown-pink precipitate.[41]

Development with ammonia gas suppresses the interference of iron. In an electrolytic attack, the latter ion is not disturbing, as the migration velocity of the copper ions is much higher than that of iron. The reaction is not very sensitive and may be recommended only for minerals that are to be attacked by strong acids.

Gold.—Even the best reactions for gold are not sensitive enough to show up this precious metal when present as an accessory element.

Benzidine, in acid solution (preferably acetic acid) gives a blue precipitate.[42] Oxidizing agents and ferric salts react equally but the latter can be easily reduced.

Iron.—Alkaline thiocyanates, in the presence of antipyrine, give insoluble red complexes with ferric iron.[43] Cobalt is disturbing, as, with the same reagent, it yields a green-blue color that cannot be masked.

Chromotropic acid in acid medium gives a direct green print with ferric salts.[44] Titanium interferes by formation of a brown precipitate.

Potassium ferrocyanide, in acid solution, yields the well known blue iron ferrocyanide. In the presence of copper (the latter ion giving a brown-red precipitate which entirely covers the iron reaction), it is advisable to develop in ammonia gas, or, better still, to use another reagent, especially chromotropic acid.

Sulphosalicylic acid gives a violet reaction with ferric iron.[45]

It will be noted that the ferrous ion has not been mentioned, although there are excellent specific reactions for it. This is because on the prints the iron oxidizes very quickly and therefore offers no problem.

Lead.—Stannous chloride and potassium iodide yield a yellow-orange double iodide with lead salts.[46] This reagent, which gives colored iodides with almost every heavy metal, can be rendered specific, since lead sulphate also reacts. The attack should be performed with a mixture of nitric and acetic acids, but if electric current is used acetic acid alone is sufficient. The excess of acid should be neutralized by ammonia gas. The paper is then slightly heated to expel the excess alkali. Finally, it is developed in the specific reagent. In the presence of interfering elements, especially copper and iron, development is first done in diluted sulphuric acid, washed in water, and treated with the reagent.

Gallocyanine[47] gives a blue reaction with lead salts in neutral solution, and with lead hydroxide, in the presence of ammonia. The reagent is not very specific, but, as it also stains lead sulphate, it can be modified to meet the desired requirement. The reaction can be performed in two different ways: (1) The paper is treated with sulphuric acid, washed in alcohol and pyridine; then it is developed in the reagent solution; (2) after the attack, the print is treated with the specific reagent, then developed in ammonia gas and thoroughly washed. A permanent blue print indicates the presence of lead (the reagent itself is blue and turns pink after alkalinization).

Dioxydiquinone and tetrahydroxyquinone both give a red precipitate with lead ions.[48] Barium and strontium yield the same reaction, but as they are not very often associated with lead, at least in sulphide ores, this reaction is useful in certain cases.

Manganese.—In order to dissolve selectively the higher oxides of manganese (pyrolusite, etc.), sulphurous acid is the most suitable attacking reagent. Under the proper conditions, benzidine is oxidized by manganese dioxide in a blue meriquinoide compound.[49] The best way to obtain a definite print is to develop the paper in ammonia gas and allow it to oxidize in the air for 5 min.; then treat it with an acetic benzidine solution, which will give a dark blue print. Another way to get a direct print is by electrolytic attack, the paper being moistened with an acetic solution of benzidine to which hydrochloric acid has been added. After the attack, the print is green-yellow and turns blue when the paper is developed in a 1 per cent caustic soda solution.

Manganese can also be identified with an ammoniacal solution of silver nitrate.[50] The print is due to a black precipitate of MnO_2 and metallic silver. However, it takes some experience to get definite results, as the iron hydroxides, which are always formed, give a brown print.

Mercury.—Aniline and stannous chloride give a black or brown spot by reduction of the mercury ion to metal.[51] Silver yields a similar reaction.

Paradimethylaminobenzylidenerhodanine gives a violet-red precipitate with mercury.[52,53] As chlorides interfere, the attack must be made with nitric acid. Copper also reacts but can be blocked by addition of sodium phosphate or phosphoric acid. The violet color, due to the presence of mercury, is perceptible even when an excess of green copper phosphate is formed. In order to get a sharp localization, it is advisable to use a green filter when taking the photomicrograph.

Molybdenum.—Potassium ethyl xanthate, in acid medium, yields a violet-red color with molybdates.[54] Unfortunately, diffusion takes place soon after the development of the print, making it necessary to take the photomicrograph almost immediately after the reaction, before the paper dries.

Phenylhydrazine,[55] which can be used instead of xanthate when molybdenum is present as an accessory element, gives, in acetic solution, a red complex. This reaction is somewhat more sensitive than the preceding one.

Nickel.—Dimethylglyoxime, in neutral, acetic and ammoniacal solution, gives an insoluble red inner complex.[56–58,18,19] Cobalt does not react, but ferrous iron yields a red color; however, if the sample is attacked by ammonia this ion cannot interfere. If hydrochloric acid is used for the same purpose, the addition of a drop of nitric acid will oxidize Fe'' to Fe'''.

When the electrographic process is used, the best dissolving reagent, aside from ammonia, is acetic acid.

Dithio-oxamide (rubeanic acid) yields a blue-violet insoluble complex.[36] This reaction is more sensitive than the preceding one, but less specific, as copper and cobalt also give prints (green for the first and tan for the second). Nevertheless, the nickel reaction covers the former one, especially if there is a slight excess of cobalt, so that only the copper ion is really detrimental.

Palladium.—Naphthalene-4-sulphonic acid-1′-azo-5-o(8)-hydroxyquinoline gives an orange-red inner complex.[59] Chromates, copper, mercuric and vanadium ions interfere to some extent.

Silver.—Para-dimethylaminobenzylidenerhodanine, in acid medium, yields a red-violet insoluble compound with silver ions.[60] The reaction is not very specific, as cuprous, gold, mercury, palladium and platinum ions are also precipitated. The above is convenient for the identification and localization of silver in silver sulphosalts. In this case, the best attacking reagent is potassium cyanide. The gelatin paper is moistened with a mixture of equal parts of 5 per cent cyanide and an acetonic solution of para-dimethylaminobenzylidenerhodanine. After contact, it is developed in dilute nitric acid and washed in acidified water. The nitric acid oxidizes Cu' to Cu'', thus suppressing this interference.

When buffered to a pH of 4, orthotolidine

yields a blue color with silver ions.[61] The attack should be made with nitric acid, and the paper dried before developing for the print.

When silver is present as an accessory element, i.e., in very small quantities (in galena, pyrite, chalcopyrite, or other minerals), an extremely sensitive reaction must be used. It is based on the fact that a mixture of a silver salt and a reducer yields a precipitate of metallic silver on the surface of silver bromide that previously has been exposed to light.[62] This reaction is more elaborate than preceding ones, but its sensitivity justifies its application over the other procedures. Nitric acid is used as an attacking reagent. The paper is then immersed in a solution of potassium bromide in order to fix the silver in the gelatin layer in the form of silver bromide; after that, to remove the last traces of the bromide, it is thoroughly washed, first in dilute nitric acid, then in water. Finally it is developed in a solution containing metol, citric acid and silver nitrate. If silver is present, a gray print will appear and slowly darken, the intensity of the color being proportional to the silver content of the ore. The developer itself at first becomes violet (formation of colloidal silver), then black, and cannot be used again. In this way the reaction is absolutely specific and, as already stated, extremely sensitive. To get a clear print, the main precaution is to use very clean reagents and wash carefully.

Sulphur.—A direct print often can be obtained by using an acid solution of antimony chloride, which, applied on the surface of sulphide minerals, yields orange antimony sulphide.[10,63] When the electrographic procedure is used, the mineral should be attacked with a weak solution of sodium hydroxide and the poles should be reversed (i.e., the mineral being connected to the negative pole). Here the reaction is specific, yielding a yellow-orange print after development in a dilute and acid solution of antimony chloride.

Sodium nitroprusside, in ammoniacal medium, gives a red-violet color with sulphides.[64] Another reaction, useful because of its sensitivity, although it does not give a print, is the catalytic effect of sulphides on a mixture of sodium azide and iodine.[65] Such a solution is perfectly stable; but in the presence of traces of sulphur ions it decomposes into sodium iodide and nitrogen. The evolution of the resulting small nitrogen bubbles can be followed easily under the microscope, thus identifying sulphide minerals.

Tin.—Cacotheline gives a red-violet reaction with stannous cations.[66] Reducers, such as antimonious and ferrous salts or sulphites, interfere but seldom are associated with tin minerals.

2-Benzlpyridine, which is green in solution, gives a red complex with stannous ions.[67] The reaction is specific, as only sulphur dioxide interferes.

Titanium.—Chromotropic acid, in acid medium, yields a brown-red precipitate with titanium salts.[68,69] The iron, which gives a violet reaction, can easily be masked by the addition of phosphoric acid.

When the electrographic procedure is employed, a direct print may be obtained by using a mixture of sulphuric acid, hydrogen peroxide and phosphoric acid as an attacking reagent.[14-16] Here, the titanium gives a complex peroxide anion, yellow in color.

Tungsten.—Tetraethylrhodamine, in hydrochloric solution, gives a red reaction product.[70] Antimony, gold, mercury and titanium interfere, if present in excess, which is very rare in minerals.

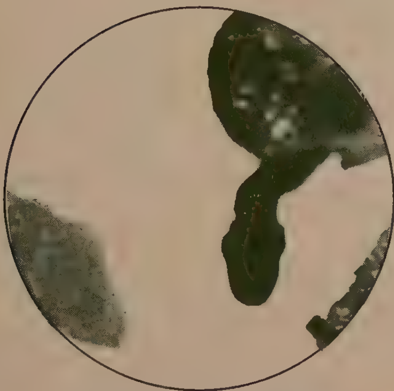
Uranium.—In neutral or acetic solution, uranium gives a brown print with ferrocyanide.[69] Ferric salts interfere, but can be masked by fluorides. Copper also has a similar reaction, but on one hand it is very rarely associated with uranium (thorbernite) and, on the other hand, it can be reduced by thiosulphate into Cu′.

Vanadium.—5-7′-dibromo-o(8)-hydroxyquinoline in strongly nitric solution (20 per cent), gives a brown precipitate with vanadium ions.[71] Iron also reacts, but it can be masked by the addition of phosphoric acid. Molybdenum yields a pale yellow color It is advisable to make a direct print by using, for the attack, a solution of the specific reagent containing 20 per cent nitric acid.

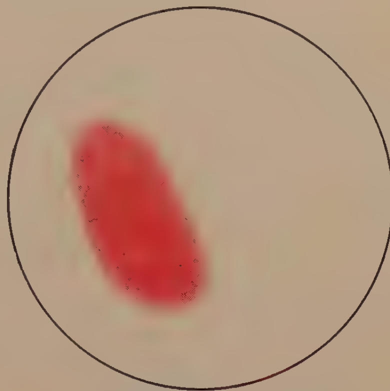
Zinc.—The double thiocyanides of mercury and alkaline metal form a blue complex compound with cobalt.[72] The precipitation is very slow, but in the presence of zinc it appears immediately. This catalytic action is the key for one of the zinc determinations. Ferric ions give a red color due to the formation of iron thiocyanide; but they can easily be masked by a rinse in ammonium fluoride solution. Copper interferes, as it gives a green precipitate.

Diphenylthiocarbazone in strongly alkaline medium (sodium hydroxide) yields a red complex with zinc ions.[73,74] Copper, mercury, and cobalt will interfere. They can be masked by adding tartrate and potassium cyanide to the reagent.

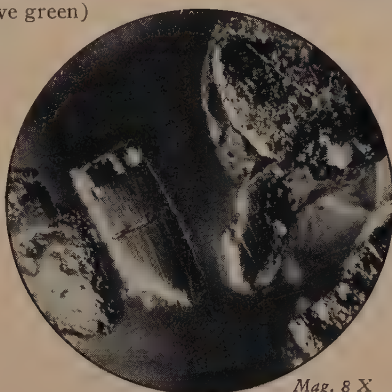
A Copper-Lead Ore Containing Native Copper, Cuprite, Chalcopyrite and Galena



Contact print A, localizing total copper (olive green) dark, native copper; medium, cuprite; light, chalcopyrite.

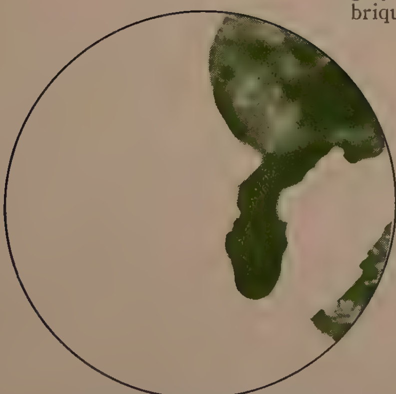


Contact print D, localizing lead—red.

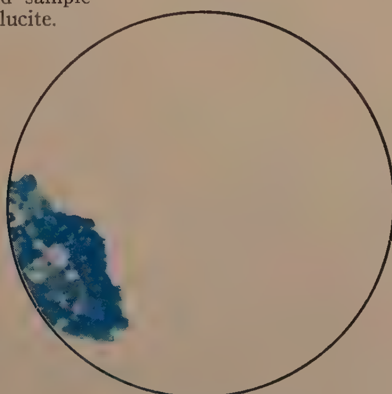


Mag. 8 X

Microphotograph of the roughly polished sample briquetted in lucite.



Contact print B, localizing copper oxide—olive green; dark, native copper; light cuprite.



Contact print C, localizing iron—deep blue.

FIGURE 1

Photomicrographic Suggestions

In order to get a definite localization of the elements, it is necessary to take photomicrographs of both the polished sample and the print and to finish them on transparent paper. By comparing and superimposing the two images, the position of the different elements can be established accurately.

It is not the purpose of this article to give instructions about photomicrographic technique, but it may be useful to offer the following recommendations:

For this purpose, it is useless to make photomicrographs with high magnification. Sixty times should be the maximum. It may be added that if some details are to be studied it is more convenient to enlarge a section from the original negative rather than to make a new one with higher powered lenses.

Keeping the same field and magnification for both the specimen and the print is important. This is accomplished best by putting the polished sample on top of a glass slide to which the print has been glued, face up; after photographing the specimen, place the slide on top of the latter, print side up, and photograph the print.

Very often filters must be used for this work in order to get better contrast, and also to mask extraneous spots due to reagents.

Results

It is evident that a method providing a means for the identification and localization of an element, as well as the determination of the form in which it is present in a given mineral, is extremely valuable both in the mineralogical field and in ore-dressing practice. To illustrate the applications of the process, some practical examples from laboratory and mill are given in the following paragraphs.

1. Identification of anisotropic sulphides, arsenides, antimonides and sulphosalts of nickel, cobalt and iron, a group that comprises the following minerals:

Nickelite: NiAs (±Sb)
Breithauptite: NiSb (±As)
Marcasite: FeS_2
Arsenopyrite-Mispickel: FeAsS
Danaite: (FeCo) AsS
Lollingite: $FeAs_2$
Safflorite-Rammelsbergite: $CoAs_2$-$NiAs_2$ (±Fe)

The hardness of these minerals is rather high (E-F of the Talmadge scale); and their color, in vertical illumination, is white (nickelite having a pink cast and breithauptite being slightly lavender). The reflecting power is strong, and varies for green radiations between 45 and 58 per cent when measured with the photometric ocular. In addition, they are anisotropic. Etching tests offer no means of distinction between them.

These minerals are difficult to attack by acids. Using contact prints, with electrolytic dissolution for certain elements, identification is easily obtained. Table 2 shows the results.

Table 2.—*Identification of Minerals*

Mineral	Reactions[a]					
	Iron	Nickel	Cobalt	Arsenic	Antimony	Sulphur
Marcasite......	+	−	−	−	−	+
Mispickel......	+	−	−	+	−	+
Danaite........	+	−	+	+	−	+
Lollingite......	+	−	−	+	−	−
Safflorite-Rammelsbergite..	o	+	+	+	−	−
Nickelite......	−	+	−	+	o	−
Breithauptite...	−	+	−	o	+	−

[a] + indicates positive; −, negative and o, weak.

Nickel: Attack with ammonia. Reaction with dimethylglyoxime.

Arsenic: Attack with ammonia and hydrogen peroxide. Reaction with silver nitrate.

Antimony: Attack with tartaric acid and phosphoric acid. Reaction with methyltrioxyfluorone.

Iron: Attack with dilute nitric acid. Reaction with potassium ferrocyanide.

Sulphur: Electrolytic attack with sodium hydroxide using inversed poles. Reaction with antimony chloride.

Cobalt: Attack with ammonium hydroxide. Reaction with dithiooxamide.

Following a similar procedure, it is possible to distinguish the very closely related individuals in the group—pyrite, cobaltite, lineites, gersdorffite, ullmanite, smaltite-chloantite.[14–16]

2. Another confusing group is the following:

Berthierite ($FeS \cdot Sb_2S_3$)
Bournonite ($2PbS \cdot Cu_2S \cdot Sb_2S_3$)
Boulangerite ($3PbS \cdot Sb_2S_3$)
Bismutite (Bi_2S_3)
Stibnite (Sb_2S_3)

These minerals are physically and optically very similar. They are soft, gray in color, more or less anisotropic and polychroic. Their reflecting power is weak to medium (about 35 per cent for the orange radiation). Contact prints permit a quick identification of each of these minerals. The technique is the following:

The mineral is attacked by a 40 per cent potassium hydroxide solution and a print is made with ferrocyanide in acid medium when looking for iron. If this reaction is positive, the mineral is berthierite; if it is negative, the attack is repeated with hydrochloric acid, and bismuth is sought. If bismuth is present, the mineral is bismutite. If that reaction is negative, the attack is repeated with 40 per cent nitric acid and the paper is developed in ammonium sulphide. An orange-yellow print indicates stibnite. A brown print shows that the mineral is boulangerite or bournonite. A positive reaction of copper with dithiooxamide decides in favor of bournonite.[10,11]

These few examples give a picture of the advantages offered by this procedure in the mineralogical field.

In the practical study of the suitable concentration process to be applied to any complex ore, the contact print method often gives the solution to the problem.

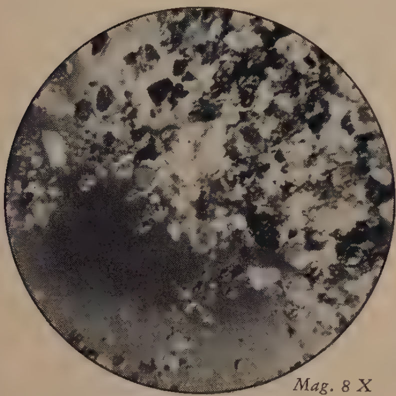
3. Flotation tests were made on silver-bearing galena in order to determine the maximum possible grade of concentration. The results being unsatisfactory, the ore was studied under the microscope and the print method was used to establish the localization of the silver. It proved that the galena contained very small inclusions of tetrahedrite, and that the silver was present only in the latter mineral. Thus, the bulk concentrate was reground and the galena selectively floated out, depressing the copper mineral by ferrocyanide. The resulting cleaner tails represented an extremely rich silver concentrate. By depressing the galena with chromate, the same results should be obtained.

4. Another somewhat similar case is that of a silver ore composed of chalcopyrite and pyrite disseminated in a quartz gangue, and showing under the microscope small inclusions of a gray and soft mineral that could not be determined by its optical and physical properties. Using the print method, it was proved to be matildite ($AgBiS_2$).[75]

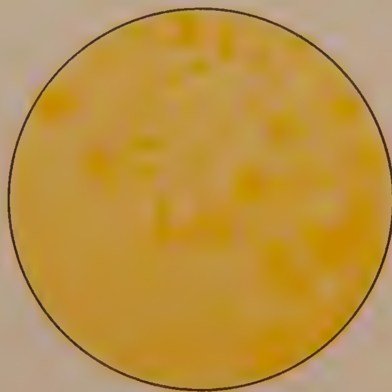
5. A silver-bearing pyrite-chalcopyrite ore was concentrated by flotation in a lime alkaline circuit. The recovery of the silver being very low, a microscopical study of the ore, using the print method, was performed. It showed the presence of two silver sulphosalts, pyrargyrite (Ag_3SbS_3) and polybasite (Ag_9SbS_6), disseminated in the chalcopyrite. However, the pyrite itself gave a clear, although weak, silver print, disclosing that part of the precious metal was contained in this iron sulphide. Therefore, flotation of the pyrite with the silver sulphosalts was indicated for good silver recovery. This was accomplished by grinding finer, using an acid circuit and depressing the chalcopyrite with ferrocyanide.

6. A company was interested in the nickel-bearing rock below their main ore body. If the metal had been contained in a definite mineral, such as a sulphide, an ore-dressing procedure could have been worked out. The print method showed that

A Complex Tungsten Ore, Containing Pyrite, Chalcopyrite, Wolframite, Scheelite, Cassiterite, Iron Oxides, etc.



Microphotograph of the roughly polished sample briquetted in lucite.

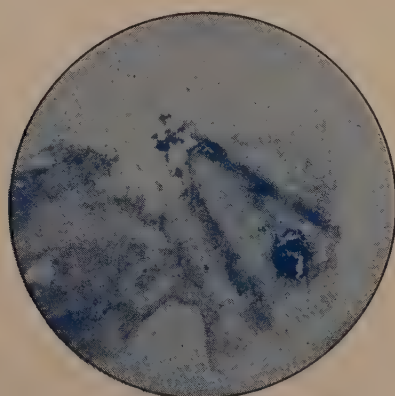


Contact print, localizing the sulfides—dark yellow.

FIGURE 2

A Manganese Ore, Containing Manganese Oxides and Silicates

Microphotograph of the roughly polished sample briquetted in lucite.



Contact print, localizing the manganese dioxides (pyrolusite)—dark blue.

FIGURE 3

the nickel was diffused through the rock itself, forming concentration zones in the most altered parts of the latter, and that any milling method other than a hydrometallurgical treatment would prove hopeless.

7. A domestic manganese ore, with a rather high content of metal, was studied in view of its concentration. Preliminary microscopical examination, completed by the use of the print method, established the fact that only a minor part of this ore was pyrolusite and higher manganese oxides, and that any attempt to recover the manganese by a physical method would be a failure.

Summary

The contact print method allows:

1. The determination of the different elements forming an opaque mineral, without altering the polished surface of the specimen.

2. The localization of these elements on the polished surface.

3. The estimation of the relative amount of a questionable element in different mineral particles, by comparing the intensity of prints obtained under similar conditions

4. The performance of a complete qualitative analysis of any metallic compound (mineral, alloy, etc.) in a very short time and without destroying the sample.

5. The solution of mineralogical and metallurgical problems, which otherwise could only be reached with very elaborate methods, and at times might even remain unsolved.

The successive operations involved by this contact print method are:

1. One surface of the ore (embedded in a plastic, or crude) is polished.

2. A contact print of this surface is made on gelatin-coated paper impregnated with a specific attacking reagent.

3. The contact print is developed in a highly selective reagent, which identifies and locates, by color, the questionable constituent.

4. Photomicrographs are taken of the polished surface and of the print, and are superimposed in order to obtain an accurate localization.

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DISCUSSION

(A. F. Taggart presiding)

A. M. Gaudin,* Cambridge, Mass.—For one who has had the privilege of corresponding for several years with the author, it is a pleasure to read in an accessible publication of a method for determining metallic minerals by a method that is specific for each ion. One of the most valuable of the features of the paper is the rather staggering list of references.

Like other methods, this one has its advantages and disadvantages. Its principal advantage, to my mind, is that it is specific to definite metallic ions when properly applied. Its principal disadvantage is that it will not reveal fine detail. The photomicrographs that accompany the paper are low-power photomicrographs: 8× is not in the same class with 800×. Yet the latter magnification more nearly represents what is customarily needed in examining minerals in flotation products where the main application of the microscope is currently made. In the oral discussion that followed the presentation of the paper, it was brought out by the author that particles as fine as 30 microns, or even finer, can be seen by his method. It is to be hoped that further advances by the author in the field of his technique will allow of sufficient refinement to make the tool that he has brought to us even more useful than it now is.

It is to be observed that, to be successful in applying the technique, one has to be rather well informed concerning the possible interfering elements, or else cook-book-recipe methods have to be adopted, with the consequent failures that result when a problem is met that is slightly outside of the ordinary.

Where minerals have distinct colors or other distinctive features, such as change in reflectivity on substitution of oil-immersion objective for air-immersion objectives, or internal reflections, it is hardly to be expected that the method of Dr. Gutzeit will recommend itself in place of the direct examination of the polished surface under the microscope. For the sulphosalts of arsenic, antimony, and bismuth, which form such a large host of gray minerals, as well as for a number of other sulphide minerals, the iridescent filming technique that was developed in my laboratory has advantages over the con-

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tact print method in that it will reveal very fine detail and in that it is a method of study that depends upon examination of the object itself, rather than of a print made from it. Frankly, I think it is superior where it can be applied.

On the other hand, for many minerals that exist in such composition as to cover a wide range of solid solutions, the contact print method, being specific to individual ions, has something that neither direct inspection under the microscope, nor the use of polarized light, nor iridescent filming can offer. In that respect its only competitor seems to be a method that has recently been developed by some of my colleagues at the Massachusetts Institute of Technology, in which a surface is sensitized by exposure in a cyclotron, with successive disentanglement of the induced radioactivity in the various atoms that constitute the surface and near surface of the specimen. That method is a contact print method also, and suffers from the same limitation that Dr. Gutzeit's method does. In conclusion, I earnestly hope that Dr. Gutzeit will continue his magnificent work and will expand the knowledge he has acquired of the reactivity of metallic minerals to their flotation treatment; no doubt many of the reagents used have value as flotation agents of one kind or another.

The Sedimentation Balance for Measurement of Size Distribution of Fine Materials

By Fred C. Bond,* Member A.I.M.E.

(New York Meeting, February 1940)

There is acute need for a method that will measure the size distribution of finely divided materials, particularly when the particle sizes are smaller than the openings of the finest screen cloth regularly available for testing. Several methods and types of apparatus are in common use, but none are entirely satisfactory for all purposes. The commonest methods are microscopic grain counts, elutriation by liquids or air, and sedimentation.

Valuable results have been secured by air elutriating devices, typified by the Haultain Infra-Sizer and the Pearson Air Analyzer. These are especially useful in obtaining sized fractions that can be examined and analyzed separately. They are particularly advantageous because the sizes of the samples produced are adequate for a considerable amount of subsequent manipulation and testing.

The sedimentation methods, such as the Wagner and Klein turbidimeters, and the sedimentation balance, do not deliver sized fractions, but they do provide a means for reasonably rapid comparative determinations of fineness and surface area, which are important in many industries.

The sedimentation balance consists essentially of a balance pan suspended in a liquid that contains falling particles. These are weighed as they settle onto the pan, and the time rate of settling is determined. The size distribution is computed from the settling rate. The balance appears to be inherently more accurate than any of the other methods, and it has served a useful purpose in the study of numerous problems, but there have been limitations which have restricted its scope of application.

Because of the essential simplicity of the method, it was felt that intensive investigation might lead to modifications that would make it a valuable tool for the operators in industries that require precise information regarding the results secured from grinding equipment. A large number of size-distribution determinations below 200 mesh were required in connection with a particular research problem in fine grinding. The sedimentation balance was used in this investigation, and a technique was gradually developed that ultimately gave good results.

A serious difficulty appeared early in the work. It was discovered that less material always settled on the sedimentation pan and more on the bottom of the containing jar than could be accounted for by calculation, and also that the material that settled to the bottom of the jar outside the edges of the pan was considerably finer than that on the pan. This segregation was so great that the distribution obtained for the finer sizes was obviously very much in error; material ground to about the degree of comminution of cement might have only 60 per cent of the calculated weight actually settling on the pan, and the error increased with the fine-

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ness of the sample. The results were often erratic, the settling rate apparently decreasing with repeated tests on the same sample.

Many reasons were advanced for this behavior, each of which led to an attempt to modify the apparatus or calculations as a corrective factor. The explanation that we accept at present is that the finely divided particles in the suspension carry an electrical charge of the same sign, hence repel each other. This mutual repulsion causes the smaller particles to move outward toward the sides of the jar as they settle, and to fall eventually to the bottom of the container rather than onto the pan. It also retards the settling rate.

The first tests were made in a large beaker, and the suspension was blunged with the sedimentation pan, according to the method used by Weinig.[1] However, it was difficult to secure check determinations by this method, especially with different operators, because of residual eddy currents and the unequal distribution of the suspended solids throughout the liquid after blunging.

In the procedure finally adopted, a sealed glass jar enclosing the sedimentation pan was swung repeatedly through an arc of 90° before weighing. This obviated most of the trouble experienced with eddy currents, and secured a more uniform distribution of the suspended solids throughout the liquid.

Some trouble was caused by incomplete dispersion, which led to the adoption of a dilute suspension. A suspension of 0.5 per cent solids, with a small amount of sodium silicate as a dispersing agent, was found to be satisfactory for the ores and other materials tested. However, in special cases the use of other suitable dispersing agents may be necessary.

When ordinary tap water was used as the suspending medium, dissolved air from the water collected on the pan and stem and affected the weight, therefore all tests were made in distilled water.

After these modifications had been adopted, another difficulty remained. Repeated determinations on the same suspension showed that the settling rate tended to become progressively slower. This was more noticeable when one determination was made immediately following another. When several hours elapsed between tests, better check determinations were obtained. However, better check determinations were obtained when 0.25 per cent of sodium chloride was dissolved in the distilled water.

Apparently, the electrical charge on the particles can be increased by increased agitation, but the addition of a suitable electrolyte equalizes or removes this additional charge, and thus establishes comparable conditions for testing.

The computations are based upon Stokes' law for the settling rates of small spheres in a viscous fluid. This law states that the constant falling velocity is directly proportional to the square of the diameter of the particle, and to the difference between the density of the particle and that of the fluid, and inversely proportional to the viscosity of the suspending medium.

It has been assumed that all of the material in the sample tested has the same specific gravity. The presence of particles with a specific gravity different from that of the average of the sample introduces a possible error in the results.

The error is unimportant where the amount of material of specific gravity different from the average value is small, as it is where the specific gravity variation is small; but when 10 per cent or more of the sample has a specific gravity twice that of the remainder it may become considerable, especially when this material tends to segregate in certain sizes.

This error is inherent in all methods of sedimentation or elutriation, since the settling rates are proportional to the differences between the densities of the particles and that of the air or liquid in which they are suspended. It is somewhat greater for the

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[1] References are at the end of the paper.

methods using a liquid medium than for those using air.

In an ore containing only a small amount of sulphide minerals it is hardly large enough to be considered, but for a heavy sulphide ore it may be important. However, no attempt has been made to apply a correction. The possibility of the error is present in all size determinations except screen analyses, and the corrected results would be misleading when compared with those made by other methods.

The particle sizes are measured in ordinal, or size, numbers.[2] According to this system, particles that pass a standard 150-mesh Tyler sieve and are retained on 200 mesh are designated as size No. 41; those that pass a 100-mesh sieve and are retained on 150 mesh are known as size 42, and so on. Thus, average particles that differ by one size number differ in diameter by 1.414, or the square root of 2, and differ in cross-sectional area by 2. The system is extended to the finer sizes, with No. 28 the beginning of the colloidal size range and No. 1 of the order of atomic dimensions.

The relationship between the particle diameter and the size number n is expressed by

$$d = a\sqrt{2^{n-1}} \qquad [1]$$

where d is the particle diameter, in microns (or thousandths of a millimeter), and a is a constant such that

$$\log_{10} a = \bar{5}.9285948 \qquad [2]$$

It is evident that a decrease of one size number doubles the settling rate of the particles according to Stokes' law. This ratio greatly simplifies the calculations of the sedimentation-balance results.

If we assume that the particles are uniformly distributed throughout the volume of the liquid, and that they fall vertically, it follows that after a given number of seconds, T, some particles of all the sizes present in the suspension will be on the pan.

If we designate as T_n the number of seconds required for a particle of size n to settle from the top of the liquid to the pan, it is evident that after T_n seconds all particles larger than size n will be found on the pan, as well as half of the particles of size $n - 1$, one-fourth of the particles of size $n - 2$, one-eighth of the particles of size $n - 3$, and so on.

In other words, if the vertical distance from the pan to the top of the liquid is 11 cm., and T_n is the number of seconds required for a particle of size n to settle 11 cm., after T_n seconds all particles of size n will be on the pan, and also all particles of size $n - 1$ that started from within 5½ cm. of the pan, all particles of size $n - 2$ that started from within 2¾ cm., and so on.

If we designate the total weight settled on the pan after T_n seconds as W_n, the cumulative weight of all material above size n is

$$\text{Cum. weight on } n = 2W_n - W_{n-1} \qquad [3]$$

This equation was derived by Weinig,[1] and can be used for calculating the percentage or cumulative weight of size n from the pan weights at T_n and T_{n-1} seconds.

It can also be shown that when the weight passing size $n + 1$ and retained on size n is designated as w_n

$$w_n = 3W_n - 2W_{n+1} - W_{n-1} \qquad [4]$$

Unfortunately, there are some possible errors involved in the assumptions underlying these equations that affect the finer sizes, and these should be analyzed.

The first of these may be called the colloidal effect. In the derivation of the equations, it is assumed that the settling ratio of ½ for each unit decrease in the size number extends to infinitely small sizes. However, the sequence is largely or completely interrupted at size 28, the inception of the colloidal range, since most, or all, of the colloidal particles are assumed to remain in suspension. This effect is inconsiderable for most of the sizes concerned, but for sizes 29 and 30 it may become important.

A second possible error is called the "buoyancy effect." If colloidal particles of size 28 are buoyed up by the Brownian movement or otherwise, with a force equal to their gravitational attraction, it is only logical to assume that particles just above the colloidal size have had their settling velocity reduced from a similar cause. Of course this possibility affects only the finer sizes.

A third possibility of error may be called the solvation effect. Since all particles are filmed with layers of water molecules of an undetermined thickness, and the attached water moves with the particle, the resultant specific gravity and settling rate of the composite water-solid particle may be lower than that of the solid alone. Like the other effects, this presumably is important only in the very fine sizes.

The presence of adsorbed air introduced large errors in the determination. However, when this air had been removed by the methods described later, the addition of various wetting agents, as well as small amounts of lime and sodium cyanide, did not materially change the results.

A fourth possibility of error is that which has been previously mentioned as caused by the electrical charge on the suspended fine particles, and it is probably more important than all of the others together. The similarly charged particles repel each other, and this mutual repulsion causes the average path of the falling particles to diverge toward the sides of the container, instead of following a vertical direction, with a consequent loss of weight on the pan. Tests have shown that this effect is important for the finer material, and is appreciable up to relatively coarse sizes. The upper limit appears to be about size 36, or particles 16 microns in diameter, although a slight repulsion has been observed at size 37. Different materials and conditions cause corresponding differences in the amount and upper limits of the error.

Many different types of apparatus and methods of procedure were tried in an attempt to obviate this error, without success. Finally the method described in this paper, in which the weights at the finer size, are

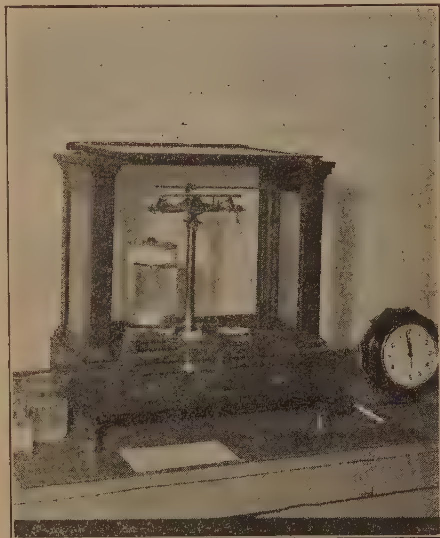

FIG. 1.—APPARATUS WITH SEDIMENTATION BALANCE.

disregarded, and the calculation of the size distribution is based upon theoretically ideal settling conditions, was developed and adopted.

APPARATUS

The apparatus used (Fig. 1) consists of an ordinary chemical balance, or any balance sensitive to one milligram or less, with the left-hand pan replaced by the sedimentation pan suspended in the sample container.

The container consists of a quart-size glass cylindrical jar with a metal screw top of the same diameter as the jar, and a cork gasket. Tops for such jars are larger in diameter than those for ordinary Mason jars, and can be obtained from the chemical supply companies. A hole 4 mm. in diameter is drilled in the exact center of the metal top, through which the stem of the sedimentation pan passes.

The pan is made from a noncorrosive metal disk 9 cm. in diameter, formed into

the desired shape. The bottom is slightly rounded, or concave upward, to allow air bubbles introduced during agitation to escape from under the pan, and a rim ½ cm. high extends around the edge. When finished, the pan is about 8 cm. in diameter, or 1 cm. less than the inside diameter of the jar.

The stem is made from a ⅛-in. stainless-steel rod, threaded at one end, and attached with bronze washers and nuts at a hole drilled in the center of the pan. The other end of the rod is shaped to hang from the balance-pan hanger on a knife-edge. A marking notch is filed on the rod at exactly 11 cm. above the pan. When finished, it is enameled and baked to aid in preventing the attachment of air bubbles to the surface.

The jar rests on a flat, U-shaped support, which clears the balance-pan arrest, with the pan suspended freely inside and about ½ cm. from the bottom. A section of a size 00 rubber stopper is placed on the stem just beneath the hanger, to prevent leakage when shaking. A second hole is drilled in the jar top and closed with a rubber stopper, for the addition or removal of water with a pipette, and the insertion of a thermometer.

An electric clock with a second hand is placed near the balance.

The average inside diameter of the jar, the depth of the salt solution in the jar, and the diameter of the sedimentation pan, are carefully measured; and the fraction of the total volume of the solution that is vertically above the sedimentation pan is computed from these measurements. This is called the Volume Fraction, and designated as *V.F.*

The instrument constant, designated as K, is 2000 divided by the Volume Fraction. The weight in milligrams of the sample taken is calculated from its density D and the constant K, so that theoretically the ultimate increase of weight on the pan should be just 2000 milligrams, from the following formula:

$$\text{Milligrams of Sample} = \frac{2000D}{V.F.(D-1)} = \frac{KD}{D-1} \qquad [5]$$

Procedure

Sufficient 0.25 per cent solution of sodium chloride in distilled water is placed in the jar, so that it just reaches the 11-cm. notch on the suspended pan stem, and a tare weight is prepared, which balances the immersed pan. The jar and contents may be weighed to facilitate addition of the correct amount of solution in subsequent tests.

It should be noted that the surface tension of the water reacting on the stem of the sedimentation pan has the effect of reducing the sensitivity of the balance, so that weighings that are accurate to less than about 3 milligrams should not be expected.

The calculated weight of the minus 200-mesh sample to be tested is placed in the jar, together with five drops of a 15 per cent solution of sodium silicate in water, and the lid is sealed on with adhesive tape, to prevent possible leakage. The suspension is blunged, or churned, with the sedimentation pan for several minutes to effect complete dispersion, and the temperature is measured, preferably in tenths of a degree centigrade.

In testing materials that tend to adsorb considerable quantities of air, it may be necessary either to heat the sample to boiling or to place it under an evacuated bell jar before making the determination.

When the sample to be tested is in suspension in water, as is the product from a wet grinding mill, and is all minus 200-mesh, it is preferable to make the test directly on the suspension, as this avoids the formation of lumps during drying. It is necessary to find the percentage of solids in the pulp, and add an amount that contains the correct number of milligrams of solid, as calculated from equation 5. Additional distilled water may then be added to bring the level up to the 11-cm. notch.

If the sample has been dried and contains lumps, these may be disintegrated by rubbing gently with a soft cork on rubberized cloth before testing.

The electric clock is set at 2 min. before an even hour and the suspension is blunged for

1 min. with the sedimentation pan. The stopper is then pressed down on the central hole in the lid, and the jar is swung vigorously through an arc of about 90°, being held horizontally at the middle of the swing. It may be rotated slightly between swings. One complete swing per second is satisfactory. The last swing should end with the jar upright just as the second and minute hands of the clock are vertical; and the jar should be immediately placed on the support in the balance case and the pan suspended and centered in the hole in the lid so as to hang freely. A suitable weight is placed on the right-hand balance pan, and the time at which it is balanced by the sediment on the pan is recorded, another weight is added and the balancing continued until a schedule extending over about 10 min. is obtained. The weights should be chosen so as to allow from about 30 to 60 sec. between successive weighings, and for convenience should be in units no smaller than 10 or 5 milligrams.

When the preliminary schedule has been obtained, the sample is reblunged and swung as before, placed in the balance, and the first weight in the schedule added. The exact time in seconds at which the balance pointer crosses zero from left to right is recorded, the second weight is added and the time of balancing recorded. This procedure is continued for 10 min., and the sample reagitated for a check determination.

Recording the time at which the pointer crosses zero gives more accurate results than attempts to measure the weight at predetermined time intervals.

Following the completion of the last 10-min. schedule, weighings may be taken at intervals as desired, and the ultimate weight obtained after settling overnight. However, the 10-min. schedule is sufficient for a complete sedimentation-balance test on samples containing particles larger than 30 microns, and the time can be extended for more finely divided materials.

Computation

The results of the sedimentation test are tabulated in three columns: (1) the weight W of the sediment settled on the pan, in milligrams, (2) the time of settling, T, in seconds, and (3) the time divided by the weight, or T/W.

The time T is plotted against T/W, and a straight line is drawn through the resulting points, and extended to zero time, in order to determine the plotted value of T/W at $T = 0$. This quantity is called b, and is used in the distribution calculation.

A convenient scale for plotting on paper 20 in. wide is obtained by using T as the abscissa with 40 sec. to the inch, and T/W as the ordinate with 0.02 divisions to the inch, both scales starting at zero in the lower left-hand corner. The T/W intercept b is found graphically.

A second line may be plotted, using both scales multiplied by 10, and a third by 100, to include the entire range down to size 28. All of the lines should have the same slope. The commonly observed upward curvature at the finer sizes is the result of mutual repulsion of the particles.

Since the object of the plot is to determine the value of the T/W intercept b, the straightest part of the plotted line is extended to the T/W axis. This is usually found in the range from 150 to 600 sec. The weight settled in the first minute or two often seems to be low, perhaps because some time is required for regular stratification of the falling particles. Repeat determinations on the same sample often appear to differ widely, but when they are plotted the values of the intercept b usually check. At least two determinations should be made on each sample to check this value.

If it is not considered necessary to plot the sedimentation results, the intercept b may be calculated directly from T and W, using the following formula:

$$b = \left(\frac{T_2T_1}{T_2 - T_1}\right) \div \left(\frac{W_2W_1}{W_2 - W_1}\right). \quad [6]$$

where W_2 and W_1 are the weights in milligrams settled onto the sedimentation pan after T_2 and T_1 sec. Of course it is not necessary to compute T/W for each weighing when this is done.

The values obtained for b are naturally more accurate when T_2 and T_1 are selected so that there is a considerable interval between them. However, values of T below about 100 sec. may give erroneous values of b, because of slow stratification of the solids.

The value of b used in the subsequent calculation of the percentage of weight retained should be the average of several computations, using different values of T_2 and T_1.

The absolute viscosity u of the water in poises may be found from the temperature by interpolation from Table 1.

TABLE 1.—*Values of Absolute Viscosity*

Temperature, Deg. C.	u, Poises	Temperature, Deg. C.	u, Poises
10	0.01308	24	0.00916
15	0.01140	25	0.00894
20	0.01005	26	0.00876
21	0.00982	27	0.00857
22	0.00960	30	0.00801
23	0.00938	35	0.00723

The time T_{36} in seconds required for a particle of size 36 to settle 11 cm. from the surface to the pan is calculated from the following formula:

$$T_{36} = \frac{115{,}500u}{D - 1} \qquad [7]$$

where D is the density (or specific gravity) of the sample.

This formula was calculated from Stokes' law for the particular conditions obtaining in the test. T_{37} is half of T_{36}, T_{38} is one-fourth, T_{35} is twice, and so on. The time for each size number can be readily calculated; and may be located on the settling-rate graph if desired, although this is not necessary for the calculation.

We have discovered, in the course of our laboratory work, that when a crushed or ground homogeneous material is screen-analyzed, the percentage of cumulative weight retained on the various sizes constitutes a hyperbolic function of the particle size. It has previously been determined by Gaudin[3] that the percentage of weight retained constitutes a logarithmic function of the size number.

The weight settled onto the sedimentation pan is a cumulative weight, although it does not correspond directly to the percentage of cumulative weight of the material; and a hyperbolic equation was derived that expresses the relationship between the settling time and the theoretical weight settled out. This equation was transposed so that the percentage of weight retained on any size number n, and passing $n + 1$, can be computed directly from it as follows:

$$\%\ \text{Wt.}_n = \frac{300(t_n)^2 b}{(t_n + b)(t_n + 2b)(t_n + 4b)} \qquad [8]$$

TABLE. 2.—*For Calculation of Surface Areas of Screen Fractions*

Mesh	n	d (microns)	Surface per Cubic Centimeter, Sq. M.
Inch			
2	60	64,360	0.0000931
1½	59	45,520	0.0001317
1	58	32,180	0.0001863
¾	57	22,760	0.0002635
½	56	16,090	0 000373
⅜	55	11,376	0.000527
Mesh			
3	54	8,052	0.000745
4	53	5,694	0.001054
6	52	4,026	0 001490
8	51	2,847	0.002108
10	50	2,013	0.002980
14	49	1,423	0.00422
20	48	1,006.5	0.00596
28	47	711.7	0.00843
35	46	503.2	0.01192
48	45	355.8	0.01686
65	44	251.6	0.02385
100	43	177.9200	0.0337
150	42	125.8112	0.0477
200	41	88.9600	0.0675
270	40	62.9056	0.0954
400	39	44.4800	0.1349
	38	31.4528	0.1908
	37	22.2400	0.2698
	36	15.7264	0.3815
	35	11.1200	0.5396
	34	7.8632	0.7630
	33	5.5600	1.079
	32	3.9316	1.526
	31	2.7800	2.158
	30	1.9658	3.052
	29	1.3900	4.316
	28	0.9829	6.104

where t_n is equal to T_n divided by 1000, and b is the T/W intercept from the sedimentation test.

A tabulation is prepared in which six successive columns are headed: n, t_n, $300(t_n)^2b$, per cent Wt., per cent Cum., and Surface, as shown in the specimen calculations.

The values of t_{36} and $300(t_{36})^2b$ are determined from the previously computed value of T_{36}, and the values for $n = 37$, 38, 39, and 40 are easily listed; since t_n decreases by ½ for each increase of one unit in the size number, and $300(t_n)^2b$ decreases by ¾. The per cent weight retained is computed from equation 8 for each of these four sizes.

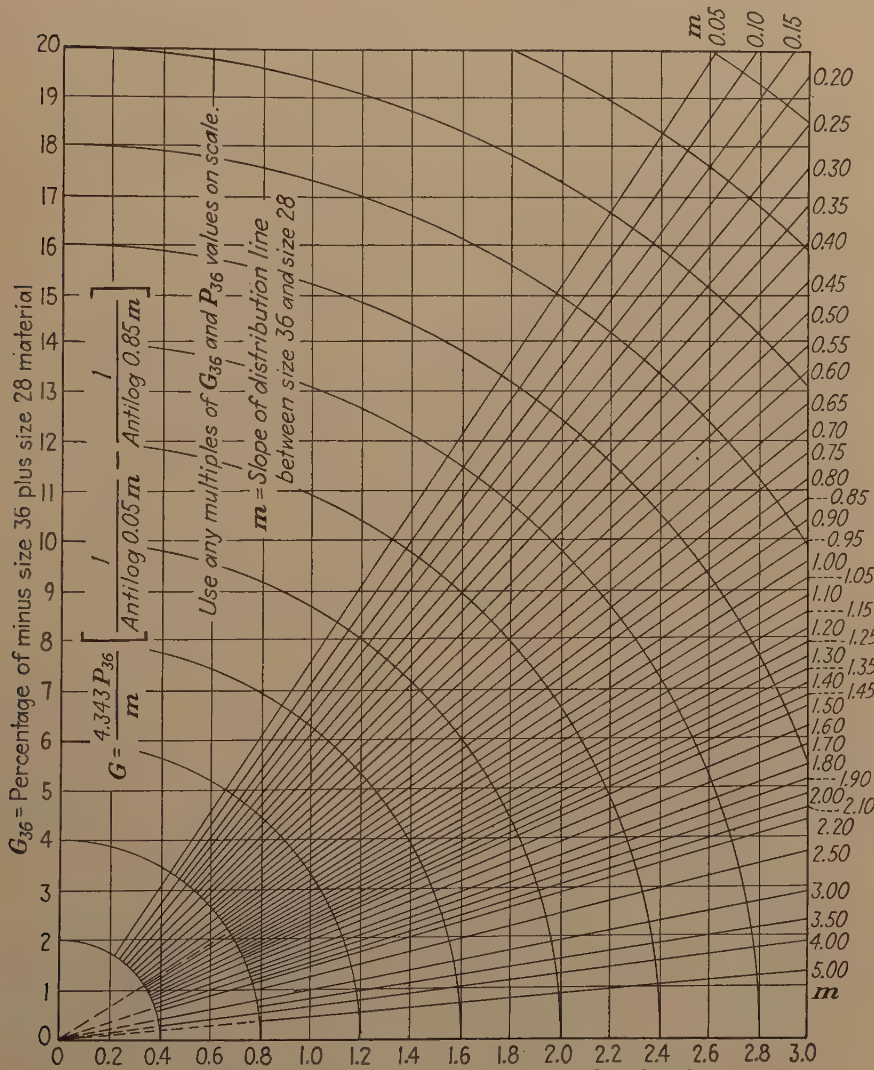

Fig. 2.—Chart for solving equation 9.

The sample should contain no plus 200-mesh material, so that the per cent weight retained on size 41 is always zero. The calculated percentage of weight retained on curve. It is always advisable to test another sample of the material on a 325-mesh sieve, and to allow for this effect in the calculated results.

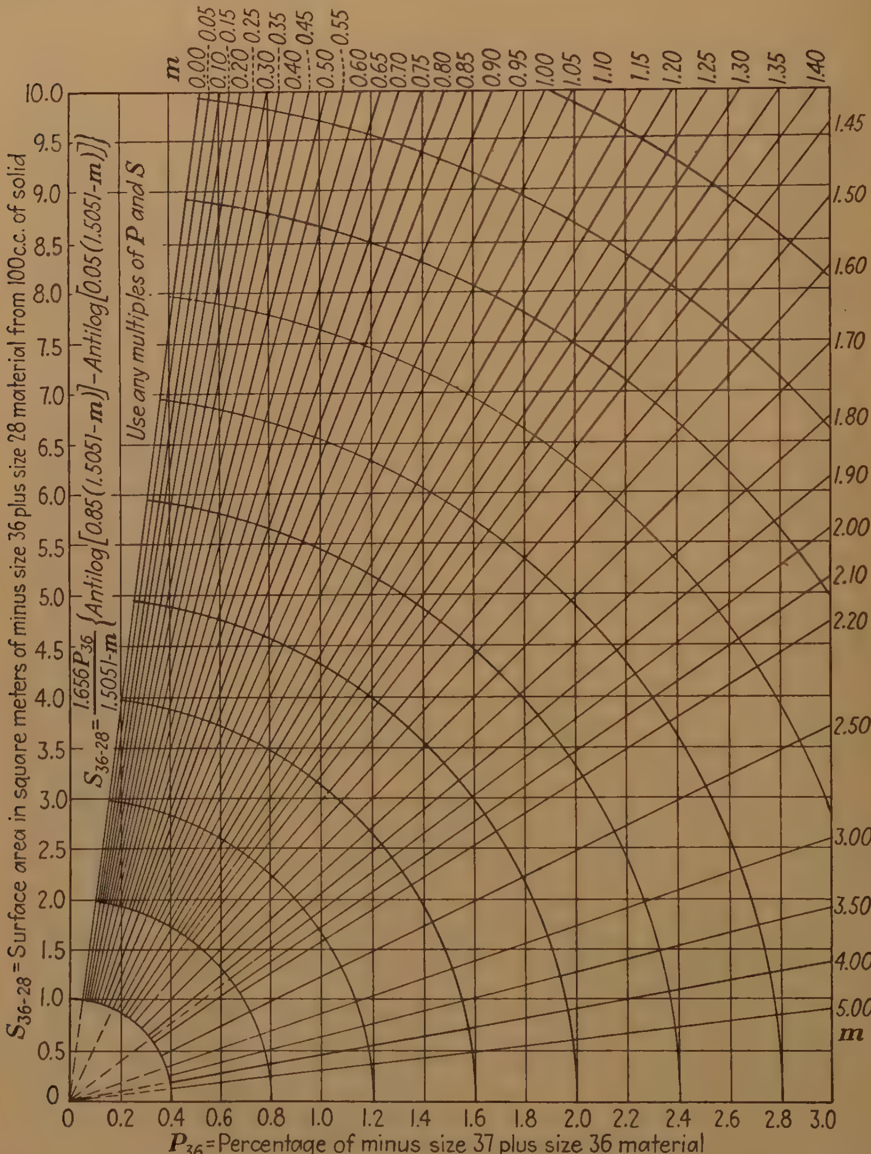

FIG. 3.—CHART FOR SOLVING EQUATION 10.

size 40 may be somewhat too large for a finely ground product, since any classifying action in the grinding circuit tends to reduce the amount of the coarsest material present below that of the theoretical distribution

It has been variously estimated that a 325-mesh sieve separates at from 60 to 53 microns. The manufacturer's calculated opening of 43 microns is obviously too small. Our tests have shown that in general a

325-mesh sieve does not retain quite all of the size 40 material. Size 40 includes particles between 52 and 74 microns in diameter.

The surface is computed for each size number by multiplying the per cent weight retained on that size by the surface in square meters per cubic centimeter of solid for that size, as listed in Table 2.

The calculation may be extended to include all sizes from 40 to 28 if desired, but the surface area and size distribution of the minus size 36 plus size 28 material may be found more quickly by a graphical method of a type previously described.[2]

Graphical Method

Let P_{36} equal the calculated percentage of weight retained on size 36 and passing size 37; and G_{36} equal the percentage of minus size 36, noncolloidal material. Also let m equal the slope of the plotted distribution line, or the logarithm of the percentage of weight retained plotted against the corresponding size number. Then

$$G_{36} = \frac{4.343P_{36}}{m}\left[\frac{1}{\text{Antilog } 0.05m} - \frac{1}{\text{Antilog } 0.85m}\right] \quad [9]$$

If S_{36} equals the surface area in square meters of the minus size 36 noncolloidal material in 100 c.c. of solid sample, then

$$S_{36} = \frac{1.656P_{36}}{1.505 - m}\{\text{Antilog}[0.85(1.505 - m)] - \text{Antilog}[0.05(1.505 - m)\} \quad [10]$$

Equation 9 was solved for a wide range of values of m and Fig. 2 was prepared, from which the equation can be solved graphically. Fig. 3 was prepared from equation 10 in a similar manner.

The value of m can be found from the known values of P_{36} and G_{36} by the use of the chart in Fig. 2, and S_{36} can be found from P_{36} and m by the use of the chart in Fig. 3. An example of the use of the charts is given in the specimen calculation that follows.

The total noncolloidal surface area of the sample is found by adding S_{36} and the surface areas as previously computed for each size from 36 to 40. There is no size 41 surface, since the samples tested contain no plus 200-mesh material.

The per cent weight distribution for the sizes from 36 to 28 can be readily computed from the known value of m if desired.

The sedimentation balance method is particularly useful in analyzing material that has been ground to nearly all through 200 mesh. It can also be used to measure the size of the minus 200-mesh portion of a coarsely crushed or ground product, but in this case the use of a 325-mesh sieve to determine the per cent weight retained on size 40 is recommended.

In analyzing cement, kerosene can be used instead of water, and the density and viscosity of the kerosene determined. If this is done, formula 7 becomes

$$T_{36} = \frac{115.500u}{D - d} \quad [11]$$

Where u is the viscosity of the kerosene in poises, and d is its density. The

Specimen Calculation

Tube-mill Product from Lake Shore Gold Mines, Ontario

D = 2.78 (density of ore)
$V.F.$ = 0.6830 (volume fraction of sedimentation balance)
K = 2000 ÷ 0.6830 = 2930 (instrument constant)
Weight of Sample = DK ÷ (D − 1) = 4570 milligrams
Temperature = 22.6° C.
u = 0.00947 poise (viscosity)
T_{36} = 614 seconds (computed by equation 7)
t_{36} = 0.614 = T_{36} ÷ 1000
b = 0.0900 (determined by plotting sedimentation results)

n	t_n	300 $(t_n)^{2b}$	Per Cent Wt.	Per Cent Cum.	Surface
41			0	0	0
40	0.0384	0.03972	1.12	1.12	0.107
39	0.0768	0.1589	8.50	9.62	1.147
38	0.1535	0.6356	15.24	24.86	2.909
37	0.307	2.5425	19.72	44.58	5.320
36	0.614	10.17	18.70	63.28	7.135
					16.618

Fourth column computed by equation 8.
Colloids in ore (minus size 28 material) = 1.50 per cent
G_{36} = 100 − 63.28 − 1.50 = 35.22 per cent
P_{36} = 18.70
m = 1.80 (from Fig. 2)
S = 42.32 (from Fig. 3)
S_T = 42.32 + 16.62 = 58.94 sq. meters per 100 c.c. solids

$$\frac{58.94 \times 100}{2.78} = 2120 \text{ sq. cm. per gram (specific surface)}$$

remainder of the calculation is made as previously described.

In the specimen calculation, the percentage of weight retained on size 40 was found

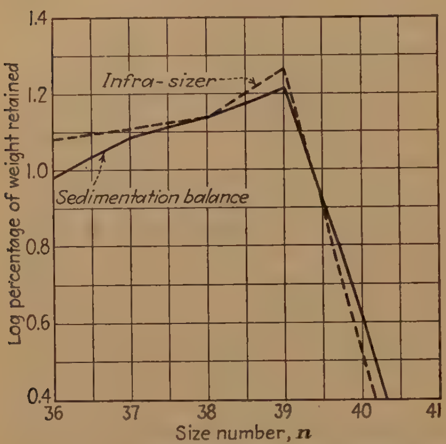

FIG. 4.—COMPARISON OF SIZE-DISTRIBUTION MEASUREMENTS ON FINELY GROUND GOLD ORE WITH THE SEDIMENTATION BALANCE AND THE HAULTAIN INFRA-SIZER.

from tests with a 325-mesh sieve. The value computed from equation 8 is 3.56 per cent, which apparently indicates that there is some selective grinding of the coarsest material in the tube mill.

A comparison is shown in Fig. 4 between the size-distribution measurements obtained on a finely ground gold ore with the sedimentation balance and those obtained with a Haultain Infra-Sizer on the same sample. The results do not differ widely.

A comparison has also been made between the surface area of a sample as computed from sedimentation-balance measurements and determinations with a Wagner turbidimeter, with results that are in fair agreement. However, much more work will be necessary before an exact and dependable correlation between the different methods of measurement can be obtained.

It should be emphasized that the mathematical treatment described was adopted to obviate the error introduced by the mutual repulsion of the suspended particles, and that it presupposes that the mat erial has a regular size-distribution curve. This can be tested in any particular case by plotting the values of T/W against T. If these do not plot linearly, neglecting the slight curvature caused at the finer sizes by mutual repulsion, they indicate either that the material is not completely dispersed or that it does not have a regular size distribution. It is then preferable to continue the balance readings down to size 28, and compute the per cent weight on each size number individually by equations 3 or 4. Such corrections for mutual repulsion can be made at the finer sizes as seem suitable from a comparison of the actual ultimate weight obtained on the balance pan, and the theoretical weight.

Our tests have indicated that finely ground siliceous ores follow a regular distribution curve, except possibly for the coarsest size of fraction present. This is equivalent to saying that they have no important natural grain size below about 325 mesh, which appears to be a reasonable assumption. However, other products, and particularly such natural materials as soils, may not follow this regular size distribution.

ACKNOWLEDGMENTS

The author wishes to acknowledge his appreciation of the comments of Mr. W. L. Maxson, and the experimental testing work of the following student engineers, all of the Allis-Chalmers Manufacturing Co., in the preparation of this paper: Bruce H. Irwin, Raymond C. Forsnas, Richard F. Thuma, John A. Fagnant, Ralph E. Cerveny, David W. Dunning.

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DISCUSSION

(*E. W. Engelmann presiding*)

R. Schuhmann, Jr.,* Cambridge, Mass.—The serious error in the sedimentation balance method described on page 300—namely, that less material settles on the pan (as much as 40 per cent less for fine material) than should do so according to calculations—probably accounts for the fact that the sedimentation balance, though given many trials, has not yet come into general use. The cause and magnitude of this error were established experimentally by Coutts and Crowther.[4] Their work showed that "The low density of suspension immediately below the pan after the sedimentation has proceeded for a few minutes inevitably sets up a flow of liquid which interferes with the free vertical fall of the particles . . . The extent of the disturbance varies with the size of particle and thus produces a distortion of the distribution curve." As a result of these currents, the space below the pan may be considered as a sort of continuous clarifier abstracting particles from suspension and recirculating the lighter "overflow" liquid.

I would hesitate to use a sizing method in which: (1) experimental errors as large as that discussed above are not eliminated or even well evaluated; and (2) the sedimentation data are not reproducible. Mr. Bond's justification seems to be that although the data are not reproducible, the value of the intercept b by his plotting method can usually be checked. Then from the value of the single constant b the complete size distribution is calculated (by Eq. 8), on the assumption that the cumulative percentage retained is some "hyperbolic" function of the particle size. Thus the usefulness of the sedimentation method described in this paper rests on the validity of the empirical Eq. 8, for which I am disappointed to find neither the theoretical nor the experimental evidence in the paper.

To save trouble for those who may want to investigate this method of sedimentation analysis, it should perhaps be pointed out that a rigorous equation for evaluating the data is:

$$\text{Cum. wgt. on } n = W_n - t\frac{dW_n}{dt}^*$$

Eqs. 3 and 4 cited by Mr. Bond are approximations (pointed out by Weinig in his original derivation) which are not close enough for many purposes.

I am in full agreement with Mr. Bond that the balance appears inherently more accurate than other sizing methods, but I believe further that the method will become useful only when the rather large errors are eliminated by proper apparatus, technique, and interpretation of data.

* Instructor in Mineral Dressing, Massachusetts Institute of Technology.

[4] Coutts and Crowther: *Trans.* Faraday Soc. (1925) **21**, 374–380.

* See *Ind. and Eng. Chem.* (1927) **19**, 60; also any of original papers of Oden, who devised the method.

F. C. Bond (author's reply).—The explanation by Coutts and Crowther for the abstraction of fine particles from the space above the balance pan is very interesting. If this is the cause of the observed error, it should be possible to design a balance pan and settling column that would give correct results directly.

The intercept b is used to avoid the necessity of making corrections for the abstraction of fine particles, and to shorten the calculations. If an irregular distribution is suspected, it can be calculated at each size number by Eq. 3 or 4, provided that a suitable correction can be made for the abstraction of fine particles.

Two references to the use of hyperbolic plotting of cumulative screen analyses are: Bureau of Reclamation, U. S. Dept. of Interior, *Tech. Mem.* No. 488 (1936), and Geco Metallurgical Bulletin, Salt Lake City (1939).

The basic hyperbolic equation used is

$$W_n = \frac{T_n}{b + \alpha T_n}$$

where α is the reciprocal of W_T. Since W_T is always 2000 mg. under the test conditions, α is always 0.0005. Eq. 8 is derived from Eq. 4 with the substitution of the numerical value of α.

The use of the rigorous equation cited by Dr. Schuhmann is, of course, limited by the intervals for which data are available. Eqs. 3 and 4 are applicable where these data are limited to successive size numbers.

The need for improvements, both in the design of the sedimentation apparatus, and in the mathematical treatment of the data obtained, is obvious.

The Law of Crushing

By John W. Bell,* Member A.I.M.E.

(New York Meeting, February 1942)

In the introduction to an excellent pamphlet, John Gross[1] makes the following statements:

> Although marked progress has been made along mechanical lines, the theory and conception of underlying principles have not advanced so rapidly. This lack of advance in theory may be attributed to the unfortunate situation resulting from the controversy as to whether the Rittinger or Kick law is applicable to crushing. Many pages of the technical literature are burdened with theoretical discussions in favor of one or the other of these laws, which have tended to cloud rather than to clear the atmosphere.
>
> Realizing the situation, the Milling Committee of the American Institute of Mining and Metallurgical Engineers, through its secretary, E. A. Hersam, studied the milling industry and, because of the apparently hopeless lack of agreement in the Kick versus Rittinger dispute (5, 13, 14, 16, 18, 19, 21, 22, 23, 24, 30, 34), concluded that a foundation for the theory of crushing was of primary importance.

The numbers in the paragraph quoted refer to papers listed by Mr. Gross in his Bibliography on Crushing and Grinding.[1] One is obliged to infer that these papers show "hopeless lack of agreement in the Kick versus Rittinger dispute." No. 22 is a paper by Arthur O. Gates[2] and No. 23 is one by the writer.[3]

Employing the subject matter contained in papers Nos. 22 and 23[2,3] the writer will show that their authors formulated identical conclusions, based solely on experimental evidence. Exactly the same defect in the Stadler hypothesis was described in each of the two papers, although entirely different experimental methods were employed in uncovering it. Both of the authors, independently, formulated the conclusion that Rittinger's was a true hypothesis and Stadler's false.

The evidence in a third paper[4] will be employed to prove that Stadler's hypothesis, when examined experimentally, is worthless.

Stadler's Hypothesis Worthless

It is the writer's opinion that the evidence available in 1915 and the corroborating evidence in 1916 was amply sufficient to eliminate the need for any further consideration of Stadler's hypothesis. Certainly the evidence produced in 1918 dealt this hypothesis its *coup de grace.* It follows, therefore, that in the year 1918 there was only one credible hypothesis left—the Rittinger hypothesis.

Gates' Opinion

What information is available in Mr. Gates' paper?[2] His conclusions are not consecutively summarized, none the less they are perfectly clear. On page 876, Gates says, and it is a very significant utterance:

> But my results show, among other things, that *many more* of Stadler's energy units are obtained per foot-pound-applied in *coarse* than in *fine* crushing. This suggests a doubt as to the correctness of his unit, since one would expect a foot-pound-applied to produce the same number of "energy units," whether the feed and resulting product were coarse or fine. When Rittinger's theory is applied, the number of mesh-tons (my unit of surface produced) is

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* Professor of Mineral Dressing, McGill University, Montreal, Quebec.

[1] References are at the end of the paper.

nearly proportional to the foot-pounds-applied, whether the product be coarse or fine; hence my conclusions favor the simple Rittinger theory that surface produced is proportional to energy applied.*

Again, on page 886, Mr. Gates states:

The following comparisons based upon the experimental work performed show that the Rittinger theory applies to crushing operations and that Kick's does not.

McGill Tests

A year later the writer's paper was published.[3] Table 1 gives a summary of the McGill results, including also a summary of the Gross and Zimmerley results. A glance at this table shows that a horsepower expended in crushing 1-in. pieces is capable of making 623, whereas a horsepower expended in crushing pieces of 0.46-in. can produce only 272 volume reductions.*

TABLE 1.—*Comparison of Tests*[a]

McGill Tests			Gross-Zimmerley Tests	
Diameter of Feed, Inches	Crushing per Horsepower: Stadler Energy Units	Crushing per Horsepower: Rittinger Surface Units	Experiment No.	Surface Produced, Sq. Cm. per Kg-cm.
1.00	623	1,198	A	19.0
0.70	472	1,192	B	16.8
0.46	272	1,097	C	18.9
0.30	190	1,002	D	18.1
0.19	150	1,115	E	18.8
0.12	109	1,054	F	19.0
0.08	82	1,028	G	17.3
0.05	77	1,137	H	17.0
0.03	77	1,250	I	16.2
Average		1,120	J	16.8
			K	16.1
			L	17.5
			M	18.2
			Average	17.56

[a] The McGill University tests show that over a wide range of diameter of piece crushed a unit of energy produces approximately a fixed amount of new surface. The results therefore confirm Rittinger's hypothesis. They also show that there is no fixed relation whatsoever between a unit of energy and the number of Stadler energy units of crushing it is capable of producing, and that far more of them are produced in coarse crushing than in fine crushing.

The Gross and Zimmerley results show that, even varying quite widely the power applied and the amount of crushing it performs, a unit of energy produces approximately a fixed amount of new surface.

Are the molecular forces resisting rupture so weak in the case of the 1-in. pieces that a horsepower is capable of producing 623 volume reductions, and do these forces suddenly become so strong in the 0.46-in. pieces that a horsepower can make only 272 volume reductions? Is not an hypothesis which suggests such variations absurd? The writer thinks it can be described in no other way. In exact agreement with Gates' findings, the McGill tests show that a Stadler efficiency number is greatly influenced by the size of the piece crushed. It is a strange characteristic of this hypothesis that invariably it yields a high efficiency if the pieces crushed are large and a low efficiency if the pieces crushed are small. Stadler's very erroneous conclusion, that a horsepower expended in stamp milling (coarse feed) produced four times as much crushing as a horsepower expended in tube milling (fine feed), is fully explained by this characteristic of his hypothesis.

Perhaps some investigators may hold the view that inability to measure surface presages inability to determine the reliability of Rittinger's hypothesis.

In the McGill tests a procedure was followed which surmounted the difficulty of inability to measure surface. This procedure was the establishment of an approximately fixed feed rate and the application of an approximately fixed amount of power. It is a reasonable assumption that, following this procedure, there would be a fairly constant amount of surface in the minus 200-mesh grade produced in each test. In other words, the surface factor might be expected to approximate a constant. Any number, therefore, such as the number adopted

* The italics have been inserted by the writer.

* Stadler computed the *amount of crushing* in terms of *volume reductions*. For some reason not clear to the writer, he called one volume reduction an energy unit.

(780), would serve provided its corresponding equivalent in terms of energy units was employed. It is noticeable, of course, that this procedure was not followed and does not apply so well in tests made on the coarser sizes (1.00 to 0.46 in.). This is not surprising when it is considered that the rolls bearings in these tests were subjected to a series of smashing shocks, whereas in the tests on finer sizes the rolls settled down to a steady grinding noise. It is only necessary to compare the variations from the mean result in the two tests to see that the McGill results make a satisfactory showing. The plus and minus variations in the McGill tests are +11.8 per cent and −10.4 per cent. In the Gross and Zimmerley tests the plus variation is 8.2 per cent and the minus variation is 8.2 per cent.

In both series of tests mechanical difficulties combine to affect the results obtained. Probably in the McGill tests variations in bearing friction mainly account for variation in result. In the Gross and Zimmerley tests, the depth of the sample crushed in the mortar would obviously require much study. The writer believes that the McGill tests clearly established the accuracy of Gates' conclusion that "the Rittinger theory applies to crushing operations and that Kick's does not."

The contrast between the lively present-day interest in crushing efficiency and the lack of interest shown in 1915 and 1916 is noteworthy and was evidenced by little discussion of the papers published at that time. Stadler feebly attacked Gates' conclusions by reiterating that his hypothesis was accepted by engineers throughout the world. Mr. Gates was naturally pleased to find that the McGill results fully corroborated his findings. Of all the members of the two Institutes, only two found enough of scientific interest in the McGill results to make comments—Charles W. Merrill and Prof. Robert Richards.* Perhaps the oddest development was that not a word of criticism of Gates' paper[2] or of mine[3] was made by any one of the theoretical supporters of Stadler's hypothesis.

Taggart's Conclusions

The third reference has to do with a paper by Arthur F. Taggart.[4] Taggart was an ardent theoretical supporter of Stadler's hypothesis based on Kick's law.[5] Loyal to his theoretical convictions, and completely disregarding the experimental proof in Gates' paper and my own, he made a series of Hardinge ball-mill tests and computed their efficiencies in terms of energy units. Guided by his Stadler efficiency figures, he formulated a number of erroneous conclusions. How dangerous it is to employ a false hypothesis and the misleading efficiency figures it supplies, to determine trends in efficiency, is clearly demonstrated as follows.

The Gates and McGill investigations show plainly that a Stadler efficiency number is greatly influenced by the size of the piece crushed. It was clear, therefore, that one would merely have to look for a Hardinge mill test on a coarse feed and another on a finer feed to expect to find the Stadler hypothesis in serious trouble. This is exactly what was found in a comparison Taggart made between Hardinge test No. 208, in which 0.96-in. feed was used, and test No. 223, in which 0.4-in. feed was employed. The important data relating to these tests are listed in Table 2.

Taggart, of course, knew that test No. 223 was more efficient than test No. 208. His third conclusion reads: "The ball-mill works more efficiently on material of intermediate (0.5 to 0.75 in. average) size than on either a coarser or a finer feed."

That observation, with respect to a coarse feed *only*, is undoubtedly correct, but in making it Taggart was obliged *to*

* Private communication.

disregard entirely his Stadler efficiency figures. They point to a conclusion diametrically opposed to the one he made.

TABLE 2.—*Comparison of Tests Made by Taggart*

Test No.	Feed Size, In.	Horsepower	Efficiency: Stadler[a]	Efficiency: Rittinger[b]	Efficiency: Minus 48-mesh[b]	Ratio of Reduction[c]
208	1	20.1	26.3	55.1	42.1	7.0
223	0.4	19.9	24.6	69.4	47.4	45.6

[a] The Stadler efficiencies show that test 208 was *more* efficient than test 223.
[b] The Rittinger and minus 48-mesh efficiencies show that test 223 was *more* efficient than test 208.
[c] The ratios of reduction show that far more crushing was done in test 223 than in test 208 and was accomplished with slightly *less* horsepower.

In the course of our discussion, Taggart remarked (p. 162 of ref. 4):

> In regard to Table 4, the writer is far from defending the mill method of using "per cent —48 mesh" or any other mesh as a measure of crushing efficiency. However such a means of measurement is used as a guide for practical work by intelligent operators of wide experience and carries weight for that reason. The near agreement reached by its use with the conclusions of the writer is not the least argument in favor of the Stadler method of measurement.

Comparison of Methods

It seemed necessary to study the results of Taggart's tests in the light of what the minus 48-mesh efficiencies might disclose. They were calculated, affording a comparison of the figures for all three methods of computing efficiency.

In Table 3, the writer has assembled the Rittinger, minus 48-mesh, and Stadler efficiency figures. The Rittinger figures are arranged in their order of increasing efficiency. The tabulation proves that with a few exceptions the minus 48-mesh efficiencies are in an order of increasing efficiency, admirably supporting Rittinger.

The Stadler efficiencies are revealed *as a collection of meaningless numbers.* There is no evidence whatsoever of steadily increasing efficiency, and so completely unsupported are they by either the Rittinger or minus 48-mesh figures that whereas these two methods show test 228 to be the test of *highest* efficiency in the quartz series of tests, the Stadler number shows test 228 to be the test of *lowest* efficiency in the same series. Surely no more evidence

TABLE 3.—*Efficiency Figures*

Quartz Test No.	Efficiency: Rittinger	Efficiency: Minus 48-mesh	Efficiency: Stadler
210	32.8	29.0	16.7
211	33.8	28.2	18.8
227	34.4	29.8	13.5
209	39.4	33.3	18.6
220	40.2	34.8	19.4
224	40.6	35.5	22.6
225	42.1	38.0	24.6
212	45.3	36.9	21.4
221	48.5	42.2	20.8
215	52.2	46.5	21.6
214	54.0	46.2	22.2
228	59.8	51.6	12.3

than this is required to prove the utter worthlessness of Stadler's hypothesis based on Kick's law.

CONFIRMATION OF RITTINGER THEORY

In 1930, Gross and Zimmerley published the complete results of their investigation,[6] and it gives the writer pleasure to be able to say that he has the most profound admiration for their contribution to the literature relating to crushing. The development of a method for measuring the surface of quartz powders was, of itself, an outstanding achievement. Their results confirm the Rittinger hypothesis in a very convincing manner, and, being able to measure surface, they were able to vary the amount of crushing considerably and still prove that the relation between energy and new surface was unchanged.

If the writer had had the conception of the existence of surface energy in 1913 that he has today, he would have felt no urge to carry out the crushing tests performed at McGill University from 1913 to 1915.

The Rittinger hypothesis can be shown quite simply to be true, since it is the only

one that satisfactorily accounts for the whereabouts of the energy actually utilized in crushing. It is well known that most of the energy delivered to a crushing or grinding machine is converted into heat, and this energy, of course, is entirely wasted. Some small part of the total energy is usefully utilized and its existence in some form must be accounted for. It can be satisfactorily accounted for only by its existence in the form of surface energy contained in the crushed material, and since the amount of surface energy locked up in the material is directly proportional to the total amount of surface, it follows that the energy utilized in crushing is proportional to the amount of *new* surface produced.

Rittinger had no conception that energy could exist in the mysterious form of surface energy. Fortuitously, however, he advanced a hypothesis that correctly represents the law of crushing; a law that may be as immutable as the law of gravity.

As will be shown later, surface energy can be liberated and caused to reappear in the form of heat.

Suggested Research

The writer would like to suggest the early execution of a research that should be vastly interesting to all who are interested in crushing problems. The objective of the research is the measurement of the surface energy of quartz. Perhaps the surface energy of magnetite and other minerals could be determined by the same method. The staff of the U. S. Bureau of Mines could readily execute the proposed research.

The method, the apparatus, and the fundamental principles have already been developed and are described in a paper by Lipsett, Johnson and Maass.[7] The fundamental principle involved is simple; i.e., that if a powdered solid having surface energy is *completely* dissolved, the equivalent of the energy reappears in the form of heat. If, therefore, the heat of solution of a sample of known surface is measured, and the heat of solution of a sample of the same weight but having far more surface is also measured, the difference between the two quantities permits calculation of the surface energy of the solid. The authors conclude that the average value of the surface energy of solid sodium chloride is 386 ergs per square centimeter at 25°.

Asked the direct question, Dr. Maass* expressed the opinion that the surface energy of quartz can be measured by dissolving it in hydrofluoric acid, employing the methods described in the paper under review. Dr. Maass would be willing to undertake the proposed research himself but, unfortunately, funds to pay for the platinum equipment that would be required are not available to him at present.

It seems clear that the main difficulty in the sodium chloride research has been the measurement of the surface in the samples treated. The proposed investigation would have the advantage that three methods are now available for estimating surface. Information regarding the accuracy or inaccuracy of these would probably be gained.

In his discussion of the Gross and Zimmerley papers, A. M. Gaudin pointed out that if very small particles were completely dissolved in the hydrofluoric acid solution the calculated surface would be low. However, it would seem possible deliberately to exclude such particles in the preparation of special samples, measure their surfaces by the Gross and Zimmerley method, or the surface meter, or by calculation, and finally employ them to measure the surface energy of quartz. Once a reliable value for this was established, measurement of the total surface energy of an extremely fine powder would be possible, and from the data its surface could be calculated in square centimeters per gram; also its average diameter. Such an investigation might supply information

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on: (1) the percentage of the total power delivered to a crusher usefully utilized in crushing quartz; (2) the screen size of the finest quartz powder produced by a crusher or grinder.

Some credence is given Edser's calculated value of the surface energy of quartz (920 ergs per sq. cm.).[8] If the surface energy of salt is 386 ergs per sq. cm., a large error in the Edser calculation seems probable. Taking into consideration the enormously greater amount of energy required to crush quartz than to crush salt, it seems hardly likely that their respective surface energies could be in the ratio of 2.4:1.

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DISCUSSION

(Charles E. Locke presiding)

J. Dasher, Jr.,* Washington, D. C.—Walker, Lewis, and McAdams, in their text, Principles of Chemical Engineering, give the law of crushing in the following form:

$$dE = -C \cdot dL/L^n$$

If $n = 1$, this equation becomes Kick's law; if $n = 2$, it becomes Rittinger's law upon integration. To me it would be just as surprising for n to equal exactly 2 as for n to equal 1.

Mr. Bell does not admit the possibility that both Kick and Rittinger may be wrong, so that he may prove Rittinger right by proving Kick, Stadler, and Taggart wrong.

Gross and Zimmerley, and others who have developed direct measurements of surface, have proved their methods only on sized samples; and at present no method—direct or statistical—is accurate enough to measure the surface of a complete comminuted product. Schuhmann[9] showed our ability to measure surface as a mountain peak with its base in the clouds, depicting our uncertainty in determining the finest sizes, which contribute most of the total surface.

It has been shown that the strength of brittle solids calculated from the lattice theory is 100 to 1000 times the observed strength of large pieces, because of the presence of flaws of varying sizes and states of development, persisting into microscopic and even molecular sizes. These fine flaws have been described by Griffin, Zwicky, and Smekal (see Houwink: Elasticity, Plasticity, and Structure of Matter). Smekal has called them *Lockerstellen.*

In the light of this knowledge, Bennett[10] quite reasonably states that breakage is determined by randomly distributed weaknesses. He proposes the following law of crushing: "When work is done on a brittle material, the energy appears in part as new surface of fragmented particles, and in part as the creation of fresh inner weaknesses."

I also agree with Bennett's statement concerning Rittinger's law, which I quote: "Rittinger's law, which states that the work of breakage is proportional to the amount of new surface produced, loses its theoretical foundation. Since the experimental evidence in favor of Rittinger's law is very slender, it is left suspended like Mohammed's coffin, unable either to take its place as a necessary consequence of the theory of matter or, on account of the impossibility of measuring with any pretence of accuracy the total surface of the broken material, to find a firm basis of experiment. It has really remained suspended in this way ever since it was first formulated by Rittinger in 1867."

Mr. Bell says: "If the writer had had the conception of the existence of surface energy in

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* Junior Chemical Engineer, Bureau of Mines, U. S. Department of the Interior. Published by permission of the Director, Bureau of Mines.

[9] R. Schuhmann, Jr.: Principles of Comminution. *Metals Tech.*, July 1940.

[10] Bennett: Broken Coal, IV. Inst. of London, 1941.

1913 that he had today he would have felt no urge to carry out the crushing tests."

I believe that if Mr. Bell had considered modern knowledge concerning the structure of solids he would have been reluctant to state that Rittinger's law might be as immutable as the law of gravity. (He did not say whether it was Newton's or Einstein's law of gravity to which he referred as the standard of immutability.)

J. W. Bell (author's reply).—I must deny Mr. Dasher's right to ascribe to me the argument contained in one of his remarks, as follows: "Mr. Bell does not admit the possibility that both Kick and Rittinger may be wrong, so that he may prove Rittinger right by proving Kick, Stadler and Taggart wrong." To me, that argument would prove exactly nothing regarding the truth of Rittinger's hypothesis. It is obvious that because Stadler's hypothesis can be proved worthless, that fact in itself offers not even a color of evidence supporting Rittinger.

Rittinger's hypothesis is proved correct to me by the fact that the energy utilized to produce crushing can only be accounted for by its transformation to surface energy. If that is true, Rittinger's hypothesis is the only one that could be correct, and I think my paper makes this argument very clear and at the same time offers a sound reason for accepting it as the only true law of crushing.

Bennett's proposed law of crushing seems to me extremely vague. If only a part of the energy usefully utilized appears in the form of surface energy, what form does the rest of it take? He suggests the creation of fresh inner weaknesses, but that does not describe a form of energy. To create a fresh inner weakness, some dislocation of the molecules would have to take place, since rock minerals are elastic, hence the creation of tiny fractures having surface energy.

Mr. Dasher is mistaken when he states that Gross and Zimmerley have proved their methods only on sized samples. I recall that Mr. Gross made a sample, by elutriation, of a quartz powder in a rising current of 0.12 mm. per second. Only the very fine particles would remain in this overflow sample. He then determined its surface and reported it to be 24,000 sq. cm. per gram.

The writer has made surface measurements employing photronic cells. When the cells were new the results were excellent. Surface measurements of 0.12 mm. per sec. overflows were found to range from 25,000 to 32,000 sq. cm. per gram.

Schuhmann has calculated the sizes in a crushed product down to particles as small as 1/1,000,000 mm. It occurs to me to wonder if it is possible to create particles of this size, taking into account the irregularities in grinding surfaces. The electron microscope might furnish evidence in respect to this.

My interest in the law of crushing was aroused by the hope that its elucidation might lead to better methods for determining crusher efficiency. While some progress has been made, a large amount of experimental work still remains to be performed.

If should be mentioned that the efficiency method developed by milling operators (tons $-x$-mesh material per hp.) reliably discloses changes in efficiency. This is quite rational, since the products of two or more machines crushing to about the same maximum size would in all probability produce approximately the same amount of surface per ton. Obviously, therefore, the test producing the maximum tonnage would be producing the most surface per horsepower.

While there may be serious errors in all the methods so far suggested to determine the surface per gram in finely ground samples, some information about these errors should be gained by execution of the research recommended in my paper.

New Units of Crusher Capacity and Crusher Efficiency

By Arthur F. Taggart,* Member A.I.M.E.

(New York Meeting, February 1941)

This paper proposes two units (believed to be new) for designating, respectively, capacity and efficiency for primary and intermediate crushers.

Capacity

Operators know that the tonnage of rock that can be put through a particular crusher in a given time depends primarily upon the crusher setting (open setting for jaw and primary gyratory types; closed setting for cones and rolls), and secondarily on the lithological character of the feed, its maximum size, the amount of fines that it carries, its moisture content, and the method of feeding. The proposed capacity formula states a relationship between all of these factors in one empirical equation, as follows:

$$T_R = TR_{80}KK'K'' \qquad [1]$$

in which T_R, called "reduction tons per hour" is the new capacity unit; T = tons of feed per hour *requiring crushing;* R_{80} = "80-per cent reduction ratio"; and K, K', and K'' are constants that approximate quantitatively lithological character, moisture content and method of feeding, respectively.

T is the hourly weight of material in the feed that is coarser than the coarsest discharge of the crusher. (For marked departures from the normal specific-gravity range—2.5 to 2.7—T should be adjusted to corresponding volumes.) In the crushers under consideration, discharge from the crushing zone is limited by the minimum dimension of a particle. Passage of the particle through a square-mesh screen is limited by its intermediate dimension. Sheppard[1] has shown that the ratio of maximum:intermediate:minimum average dimensions of broken rock fragments may be taken as about 1.5:1:0.6 for nonslabby rocks such as are normally crushed, while for slabby rocks, such as thin-bedded sedimentaries, foliated schists and gneisses, and the blocky metamorphics, the ratio is of the average order 1.3:1:0.3, with the possibility of an even smaller relative value for the thickness. Gaudin[2] finds the corresponding ratio for −150 + 200-mesh pyrite to be 1.4:1:0.6. The hourly tonnage of crusher-plant feed, therefore, should be diminished by the amount thereof that would pass a square-mesh screen of which the aperture is S times the crusher setting, where S is a shape factor with the value 1.7 for rocks that break exceptionally cubic; 2 for ores and rocks that exhibit no appreciable tendency to slabbiness; and 2.5 to 3, or, in exceptional cases, even larger, when the tendency to slabbiness is marked.

R_{80} is the quotient of the theoretical square-mesh aperture that would pass 80 per cent of the feed, divided by the aperture that would pass 80 per cent of the product. It is best determined from screen tests.

If, as is usual with primary crushers, a sizing test of feed is not available, the

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[1] References are at the end of the paper.

80 per cent size thereof (l_F) may be estimated* with sufficient precision from

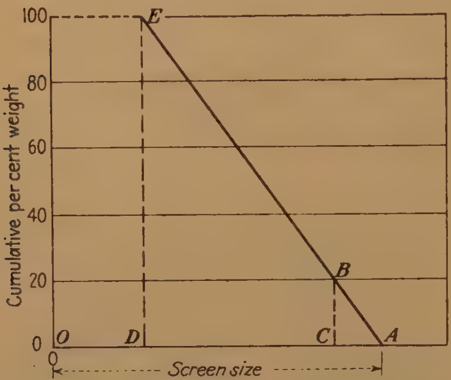

FIG. 1.—GRAPH OF FEED TO PRIMARY CRUSHERS.

the aperture of the feed-guard grizzly as follows:

$$l_F = \frac{S(4g + s_0)}{5} \qquad [2]$$

If the sizing curve of the product of a primary crusher is unknown, the 80-per cent size (l_P) may be estimated on the basis

$$l_P = Ms_0 \qquad [3]$$

where M has the values given in Table 1.

The reasons for using R_{80} instead of R_{Limiting} or $R = g/s_0$ is that R_{Limiting} may be <1 for slabby feeds, that values for g are not available, and those for s_0 are not significant in intermediate crushers.

Choice of the 80-per cent point is based on the fact that the effect of stray coarse material on the curvature of the sizing-test graph has disappeared at that point, while

TABLE 1.—*Values of M in Equation* 3

Character of Feed	Type of Crusher	M
Easily crushed rock, unscalped......	Jaw	0.90
	Gyratory	0.75
Easily crushed rock, scalped........	Jaw	1.15
	Gyratory	1.00
Average rock, unscalped...........	Jaw	1.15
	Gyratory	1.00
Medium tough or slabby rock, unscalped........................	Jaw	1.40
	Gyratory	1.25
Average rock, scalped...............	Jaw	1.40
	Gyratory	1.25

the remaining segments of the sizing curves still have a substantially maximum spread.

K is 1.0 for rocks such as medium hard to hard limestones and the like. When crushing harder, but clean-breaking stone, such as the general run of undecomposed igneous rocks, the same factor may be used provided the crusher is massive enough to stand up under the strain of such overpowering as is necessary to prevent slowing down under full load. For crushers not so sturdy, K should be taken as 0.85 for rocks of the general crushing behavior of hard, tough granites, and 0.75 for the tough basic igneous rocks such as traps and diabases. If rock is not clean-breaking, whether harder or softer than limestone, or if it tends to shatter on initial break to produce a choking tendency in the crushing zone, K should be taken as less than 1.0; probably around 0.75 is safe, unless the condition is aggravated.

K' will normally be 1.0, but may fall to 0.5 or less with crushers of the gyratory type, either primary or secondary, when set for relatively fine crushing, if undersize is fed containing just the intermediate amount of moisture that causes the fines to cake more or less when squeezed in the hand.

K'' will rarely exceed 0.75 for primary crushers, and probably averages nearer 0.5, although data for close quantification are lacking. Secondary crushers fed from surge

* The basis of estimate is that the feed to primary crushers (run-of-mine or run-of-quarry) approximates straight-line distribution in the coarser sizes. The graph of such a feed, scalped to the crusher open setting, is $\overline{AE}$ in Fig. 1, where $\overline{OA} = Sg$, when g = aperture of the grizzly limiting maximum size fed to the crusher; and $\overline{OD} = Ss_0$, where s_0 is the open setting of the crusher; S being the shape factor. Then

$$\overline{OC} = Sg - \overline{AC} = l_F$$

and, by similar triangles, and substitution as above, equation 2 develops.

bins by accurately controllable feeders can be operated with $K'' = 0.9$ to 1.0, but such feeding is rare in mill practice.

The product $KK'K''$ may be approximated in most cases by comparing T_R in average mill performance with values derived from the manufacturer's rating for the same machine. Manufacturers' ratings are usually based on thick-bedded, moderately hard, blocky limestone ($K = 1$); limited as to upper size so that it will enter the crusher and nip without delay due to bridging or nip failure, and scalped to remove all material finer than crusher open setting ($K' = 1.0$); fed by an attendant at the maximum rate—i.e., so that the crushing zone is kept full at all times ($K'' = 1.0$). The best data at hand indicate that for jaw crushers values of $KK'K''$ in the mills range from 0.3 to 1.8 times the values based on the manufacturer's ratings, averaging 1.0 for 20 installations; while for gyratory crushers the range is 0.3 to 2.3, averaging 0.88 for 17 reports. Two out of three field gyratories show values of $KK'K''$ below 1.0, the discrepancy being greater for the large crushers. So far as it is possible to evaluate the large discrepancies, the low field performances correspond to crushers that are oversize because of reception demands, while the high apparent performances are due to crediting the crusher with finished size in the feed, whether this was actually scalped or not.

Efficiency

Efficiency, in the usual mechanical sense of the ratio of work output to energy input, is not determinate for crushing operations, because of our inability to measure work output. Attempts to estimate output in terms of an equivalent unit, based on screen analyses, have been made,[3] but despite considerable experimental investigation and a large amount of contentious writing, no useful method has evolved. Millmen now, therefore, make a statement of effectiveness in terms of "tons per hp-hr. (or kw-hr)," with, usually, an accompanying statement of the limiting sizes of feed and product. The inadequacy of such a state-

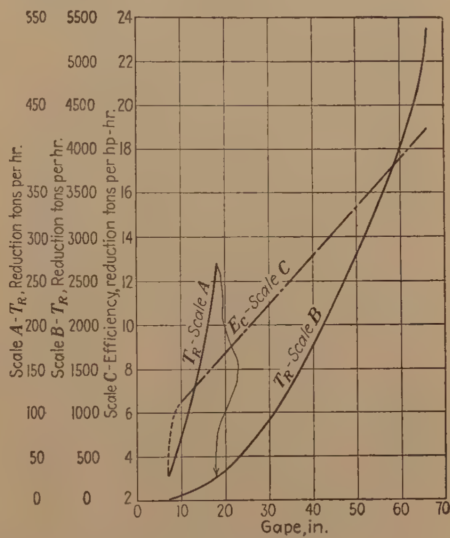

FIG. 2.—CAPACITY AND EFFICIENCY OF STANDARD JAW CRUSHERS.

ment is immediately apparent upon examination of a table of estimated crusher performances such as is found in any manufacturer's catalogue. Thus, for example, a 15 by 24-in. Blake crusher is rated for 15 to 33 tons per hour at settings of 1.5 to 3 in., respectively, at one power draft, say 30 hp., and will show therefore from 0.5 to 1.1 tons per hp-hr., according to the setting. Yet the same machine is actually doing substantially the same amount of effective primary crushing per unit of energy input throughout this range of tonnages and settings, and should, if the proper unit is chosen, show the same efficiency.

The efficiency unit proposed—using the word efficiency in the broad sense of relative effectiveness—is the "reduction ton per hp-hr." (E_c). It is derived from the equation:

$$E_c = \frac{T_R}{P} \qquad [4]$$

where P = horsepower "consumed," which is actually energy input, since the time factor is included in T_R.

Values of E_c computed from manufacturer's catalogue data are substantially constant for a given crusher throughout its range of settings (for the 15 by 24-in. crusher cited above the values are 4.8 and 5.4 for the respective instances); they increase for a given type of crusher with increase in size of machine (Fig. 2); they are lower for the jaw than for the gyratory over the same feed-size range; and are higher for cone-type or high-speed gyratory crushers than for jaw or primary slow-speed gyratories in the fine-product sizes; all of which facts are well in accord with operators' experience.

Fig. 2 gives values of T_R and E_c for jaw crushers with standard straight-element plates, according to manufacturer's ratings, adjusted to a ratio of length of receiving opening L to gape G of 1.5. If values are desired for crushers of different L/G ratios, the values of T_R as read from the chart should be multiplied by the quantity $\frac{L}{1.5G}$. Values of E_c are the same irrespective of L/G, for the reason that power consumption for crushers of the same gape and different lengths of receiving opening is subject to the same adjustment as T_R and the value of the ratio T_R/P is not, of course, affected when numerator and denominator are both multiplied by the same factor.

In calculating the values for plotting Fig. 2, average values for P taken from manufacturer's catalogues were multiplied by the factor 0.85. This is an average factor from some 20 to 25 reports from the field on power drafts by fully loaded jaw crushers.

The use of Fig. 2 for estimating can best be explained by an example. Assume that a crusher is wanted to break 50 tons per hour of run-of-mine granite that has passed through an 18-in. grizzly. It is desired to have a product that will pass a 4½-in. grizzly.

T

Normal run-of-mine granite will break roughly to a straight-line cumulative curve. Hence the tons of −4½-in. fines in the feed will be 12.5 tons per hour, leaving a net scalped feed of 37.5 tons per hour.

K for granite = 0.85

K' for scalped feed = 1.0

K'' may be taken as 0.75 on the assumption of a headframe surge bin with pan feeder on push-button control. Then

$$T = \frac{37.5}{0.85 \times 0.75} = 59$$

R

Feed sledged through an 18-in. grizzly may be taken to have 18-in. thickness of largest particle.

The square-mesh size (intermediate dimension) of such material will average about $\frac{18}{0.6} = 30$ in. This sets a minimum length of receiving opening, if sledging at the crusher is to be avoided.

If the product is to pass a 4½-in. grizzly, the *open* setting should be 4 to 4¼ in., say 4. The square-mesh limiting screen for the product will be about $4 \times 1.7 = 6.8$ inches.

The 80-per cent point of the feed, from equation 2, is 25.8 in. The 80-per cent point for product, from equation 3 and Table 1, is 5.6. Hence

$$R_{80} = \frac{25.8}{5.6} = 4.6$$

and

$$T_R = 59 \times 4.6 = 272$$

From Fig. 2, the corresponding gape is 18 inches.

A crusher to receive feed of 18-in. maximum thickness should have $G = \frac{18}{0.85}$

= 21.4-in. gape, in order to assure ready nip.

The receiving opening required is, therefore, 21.4 × 30-in. minimum. A 24-in. gape is the nearest larger standard. A 24 × 30-in. straight-element crusher will serve, but if a small safety factor for later plant expansion can be afforded, a 24 × 36-in. machine should be chosen to eliminate danger of necessity for sledging to permit entry of particles with particularly long intermediate dimensions.

P may be determined from equation 4. E_c from Fig. 2 for a 24 by 36-in. crusher is 9.6. $T_R = 272$. Hence

$$P = \frac{T_R}{E_c} = \frac{272}{9.6} = 28.3$$

Since Fig. 2 is plotted on the basis of 85 per cent of operating peak-load power draft, the minimum motor rating required = 28.3 ÷ 0.85 = 33.3 hp. The crusher chosen, however, was a 24 by 36, which has a reduction-ton per hour capacity of 575. The motor rating required for this is $\frac{575}{0.85 \times 9.6} = 70.4$ hp. Good practice would be to install a 75-hp. motor to permit of cold starting without excessive overload, and to give sufficient power if it should be desired to work the larger crusher at full capacity for a shorter daily period.

Similar curves can be drawn for other types of crushers, making suitable changes in the method of calculation to adapt them to the particular type of crusher.

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Dust Control in Large-scale Ore-concentrating Operations

By Robert T. Pring,* Member A.I.M.E.

(New York Meeting, February 1940)

In addition to the humanitarian aspects of a dust-control program, certain economic benefits are becoming more fully recognized and now furnish a greater incentive to the mill operator to eliminate the dust from his plant. It is well known that a favorable working environment promotes increased efficiency and improves the morale of employees, with a consequent decrease in labor turnover rates. One progressive mining company[1] estimates its direct and indirect labor turnover costs at from $40 to $50 per man; for skilled labor, such as machine operators, the cost may approach $100 per man. While these figures may not be typical of every plant, they do indicate a potential savings in operating costs.

Dust signifies waste; its *elimination* decreases maintenance costs of mechanical and electrical equipment; its *recovery* is often justified from the standpoint of values contained in the collected material, although in large-scale milling operations handling low-grade ore this may not be an important factor.

Perhaps the most significant trend in industrial sanitation is that dust-control equipment and procedures are becoming recognized as integral parts of the mill flowsheet. Exhaust ventilating systems are now afforded equal care in design and maintenance to that given other machinery more closely related to the process.

Methods of Dust Control

"Dust control" is a general term descriptive of a number of methods used singly or together to prevent particulate atmospheric contamination. These methods are, in order of their desirability, as follows:

1. Change of process.
2. Isolation of process.
3. Wet methods.
4. Ventilation.
   a. Local exhaust ventilation.
   b. General ventilation.
5. Personal respiratory protection.

In ore-concentration plants it is seldom practical to change a process to eliminate the dust it creates It is possible in some cases to isolate a particular dust-producing operation from the rest of the plant, provided that constant attention on the part of the operators is not required. Frequently water is used in fine crushing and fine grinding, primarily to facilitate the processing of the ore and incidentally to allay the dust produced. Wet methods, however, are not feasible in coarse and secondary crushing operations, because of the relatively great volume of water required in instantly wetting each freshly fractured rock surface. Such quantities of water would entail conveying and storage difficulties. Local exhaust ventilation is the most effective and most commonly applied control measure, wherein directional air currents through the zone of dust production draw the contaminants into a duct system.

Unfortunately, only a limited amount of reliable information on exhaust ventilation is available.[2-5] Of necessity these publications are rather general, being primarily concerned with engineering fundamentals. While many organizations have successfully

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* Utah Copper Co., Salt Lake City, Utah.

[1] References are at the end of the paper.

solved their dust problems, the valuable data accumulated during the installation and operation of their dust-control equipment are too often buried in confidential files. Without the aid of extensive field work it is inevitable that the ventilation engineer must base his design on certain assumptions. Only by the appraisal of a number of successful exhaust systems and the tabulation of pertinent data covering a wide range of conditions can the necessity for these assumptions be minimized.

Ventilation at Arthur and Magna Concentrators

Since 1936, more than 23 centrifugal fans, handling in excess of 305,000 cu. ft. per min. of air, have been installed to control the dispersion of dust in the Arthur and Magna mills, with units totaling an additional 100,000 cu. ft. per min. remaining to be completed in 1940. Of these installations, one of the largest is described in some detail in the following pages, to demonstrate methods used in design.

The Arthur and Magna mills have a combined capacity of 66,000 tons per day during normal operation and a total payroll of about 1700 men. The ore milled is received in 80-ton and 100-ton ore cars from the company's opencut mine in Bingham Canyon. It is a low-grade monzonite porphyry[6] containing 2 to 3 per cent of sulphide copper minerals, mainly chalcopyrite, chalcocite, bornite and covellite, and, in addition, small amounts of silver, gold and molybdenite. The gangue minerals consist principally of orthoclase feldspar, albite, biotite, chlorite, and quartz. The moisture content of the ore averages from roughly 4.5 per cent to not more than 7 per cent. Fragment size and hardness as well as mineralogical composition may vary from day to day. The apparent density of the dry material averages 170 lb. per cu. foot.

Based on a study of the composition of air-borne dust and on an extensive program of physical examinations, a maximum concentration of 15 million particles of dust per cubic foot of air was established as the goal to be attained. In view of the possibility of state codes governing permissible dustiness, every effort was made to further reduce the concentration to a figure well below that which any reasonable state code might require.

Exhaust ventilation of large ore-concentration plants present problems common to few other industries and necessitates careful analysis of the individual requirements of each plant. In the first place, the conveying of material plays a prominent part in the mill flowsheet, and, for reasons of economy, must utilize gravity to the fullest extent. To date, no workable method has been developed for the computation of rates of air flow induced by falling ore. The number of variables is great and the part played by each is difficult to estimate.

In most manufacturing operations, dust produced is frequently a valueless by-product and local exhaust hoods are designed to carry all of the contaminating material into the duct system by means of high-velocity air currents. In ore-crushing plants, on the other hand, the dust formed is part of the product itself and exhaust ventilation seeks only to prevent its dispersion into the surrounding atmosphere. Reasonably airtight enclosures, maintained under slight negative pressure, are effective in accomplishing this end and can be designed to drop out the coarse material before it enters the exhaust piping.

At the Arthur and Magna concentrators certain factors influence the design of dust-control equipment. Because of the large tonnages handled and the size of surge bins, together with the vertical distances through which the ore falls, any effective exhaust ventilating installation itself becomes a large-scale operation; therefore a reasonable amount of field experimentation is justified in the increased efficiency and decreased cost of the completed installations.

Ventilation of Secondary Crushing Department at Magna Mill

Of the exhaust ventilating systems now in operation at both the Arthur and Magna mills, the installation in the secondary crushing department of the latter was selected for discussion because the problems involved were more or less typical of large ore-concentration plants. Fig. 1 is a plan view and Fig. 2 a sectional elevation of this plant. From the primary crushing department the ore is transported by two 60-in. inclined conveyors (1) and discharged at the top of the secondary crushing plant tower into a distributing bin (2) from which it is fed to four sets of grizzly-screen assemblies (3) grouped symmetrically about the tower axis. The grizzly oversize falls through chutes to the feed hoppers (4) of four Symons 7-ft. cone crushers (5), the undersize passing over impact screens. The screen oversize also passes into the crushers, while the undersize drops almost vertically at (6) through a large central surge bin (7) through two ore drops (8) on to two 54-in. belt conveyors (9) under the bin. The product of the crushing operations joins the screen undersize in the surge bin through (10). The bin is 26 ft. high and 10,080 cu. ft. in volume. The total fall of the ore from head pulleys to the conveyors below is 60 ft., with a *free* drop inside the bin of 11 ft. for the crusher discharge and 26 ft. for the screen undersize. The total tonnage through the plant may run as high as 1700 per hour.

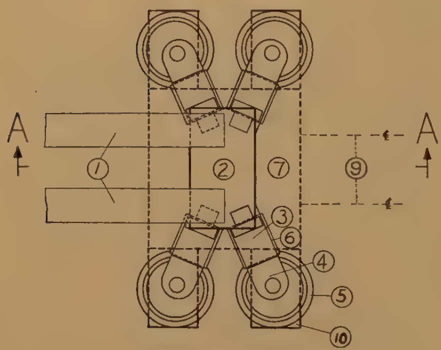

Fig. 1.

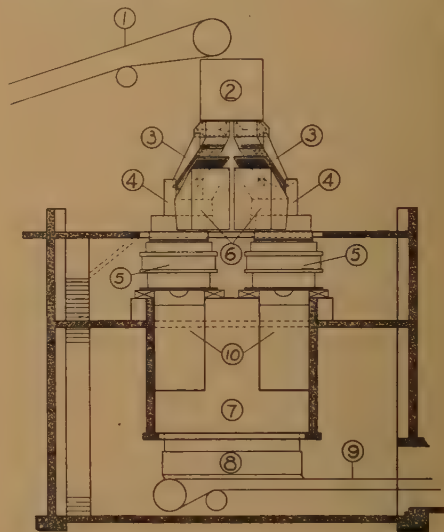

Fig. 2.

Figs. 1 and 2.—Plan view (1) and sectional elevation (2) of secondary crushing plant at Magna mill.

Falling ore acts as a fan, pulling air in above and forcing it out below. Since air leaking out must first get in, properly constructed enclosures greatly reduce the volume of dusty air forced out below. Consequently, the first step was to enclose the entire process as completely as possible. Skirted hoods were installed over the head pulleys of the primary inclined conveyors, all chutes and hoppers were repaired and covered, and the bins were rebuilt of two thicknesses of wood with tar paper between. The enclosures on grizzlies, screens and crushers were not completely satisfactory because of the necessity of frequent inspection and repairs at these points.

The frequent use of the overhead crane and the fact that the crushing-plant tower was situated in the center of the building made it impossible to install hoods locally

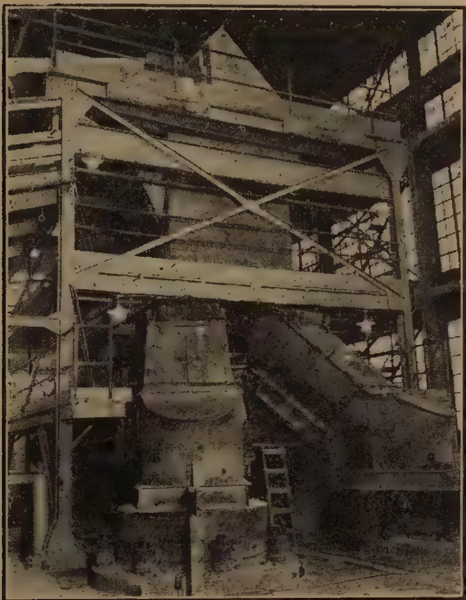

FIG. 3.—TOWER IN SECONDARY CRUSHING PLANT.
Showing method of enclosing impact screens, grizzlies and crushers.

on crushers, hoppers, etc. (Fig. 3). It was necessary to evacuate enough air from the surge bin to maintain a negative pressure throughout the tower. Ore drops could if necessary be hooded and exhausted separately.

Through the openings not amenable to sealing, air currents were noted as follows: inward at primary conveyor head pulleys, distributing bin, and at the rear of the grizzly-screen units; outward elsewhere about grizzlies and screens, crusher hoppers, crushers, surge bin and ore drops. Quantitative measurements of escaping air were not reliable above the surge bin. To determine the total air-flow requirements, it was necessary to measure the volume of air escaping from the bin (with all doors opened) and ore drops, adding to this figure a reasonable amount which would bring the bin under negative pressure and create inward air currents from the head pulleys down to the bin.

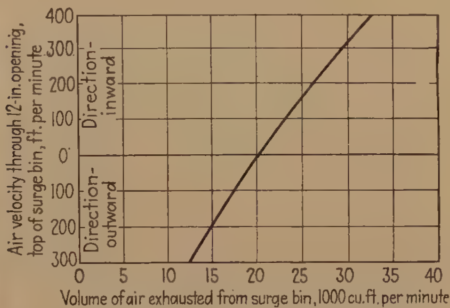

FIG. 4.—RELATIONSHIP OF AIR DRAWN FROM SURGE BIN TO VELOCITY OF AIR CURRENTS LEAKING OUT.

Anemometer readings at four open doors in the surge bin and at the front of the ore drops showed: (1) an average velocity of 305 ft. per min. through 59.7 sq. ft. of opening in the surge bin, or 18,200 cu. ft. per min. of air escaping; (2) an average air velocity of 1130 ft. per min. from the open front of each ore drop corresponding to an air-flow rate of 6400 cu. ft. per min. At the same time air currents ranging up to 376 ft. per min. were measured escaping from chutes, hoppers and crushers.

Because of the lack of space for hoods and piping in the conveyor tunnel, it would have been advantageous, from the standpoint of operation and repairs, to create an inward air flow at the ore drops by means of a single-pipe system in the surge bin. While the success of such a venture was doubtful, it was decided to investigate its feasibility by installing an experimental unit. Accordingly two 40-in. salvaged fans were connected to the surge bin by means of salvaged light-gauge piping. The direction and velocity of air currents passing through a 12 by 12-in. opening in the top of the bin were determined for different rates of flow through the experimental fan system. Fig. 4 shows the relationship of these air currents to the volume of air drawn from the bin.

When 20,300 cu. ft. per min. was exhausted by the fans, no air motion was detected through the test opening, although air currents escaping from grizzlies, screens, and hoppers were not influenced.

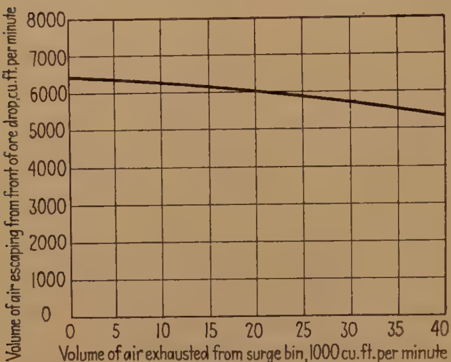

FIG. 5.—RELATIONSHIP OF AIR DRAWN FROM SURGE BIN TO VOLUME OF AIR LEAKING FROM FRONT OF ORE DROPS.

Little change could be measured in outward air velocities at the ore drops under the surge bin. Fig. 5 shows the relationship of the volume of air leaking from the ore drops to the volume of air pulled from the surge bin, and indicates that separate hoods were required at those points.

Carrying the experimentation farther, the fan capacity was increased until positive inward air currents were created throughout the entire superstructure of the tower. Table 1 shows the total volume of air exhausted from the surge bin to produce inward air currents of 100 ft. per min. through the various crusher feed hoppers. These data indicate that although 20,300 cu. ft. per min. of air drawn from the surge bin was sufficient to prevent outward leakage

TABLE 1.—*Volume of Air Exhausted from Surge Bin to Produce Inward Air Currents of 100 Feet per Minute through Crusher Feed Hoppers*

LOCATION OF FEED HOPPER	REQUIRED VOLUME CU. FT. PER M.N.
No. 1 crusher	28,000
No. 2 crusher	26,000
No. 3 crusher	32,400
No. 4 crusher	34,500

through the top of the bin, nearly 35,000 cu. ft. per min. was required to produce desired results at points above the bin. It would have been more economical to exhaust each crusher locally, but, in this case, the facilitation of operation and repairs prevented the location of any piping above the operating floor.

Summarizing the observations described above, 18,200 cu. ft. per min. of air was measured escaping through openings in the surge bin, but 20,300 cu. ft. per min. drawn out by means of fans was just sufficient to counteract outward leakage at the top, an increase of 11.5 per cent. Roughly, 35,000 cu. ft. per min. was required to control air currents above the operating floor, or 190 per cent of that measured through the open doors of the bin. At the front of the ore drops the average of all readings showed that 6400 cu. ft. per min. was escaping. In addition, some air leakage was detected at the rear of the ore drops, near the tail pulleys of the 54-in. conveyors.

The difference of 11.5 per cent between measured air currents from the surge bin and values obtained with the experimental

TABLE 2.—*Influence of Area of Test Openings to Ratio of Measured to Actual Air-flow Requirements*

Area of Test Opening, Sq. Ft.	Air Velocity through Opening, Ft. per Min.	Measured Air Flow, Cu. Ft. per Min.	Measured Air Flow / Required Air Flow
15	490	7,350	0.36
30	332	9,960	0.49
59.7	305	18,200	0.89

fan system indicated that not less than 15 per cent should be added to similarly determined air-flow requirements. Further testing, however, served to demonstrate that the relative size of the openings in the bin through which escaping air currents were measured greatly affected the magnitude of the percentage correction. Table 2, based on limited data applying to this one surge

bin only, shows the influence of the area of test openings on the ratio of measured to actual air-flow requirements, where the plane of the test openings was parallel to the path of the falling ore.* Generalization on these ratios is not possible except as follows:

1. Where the ratio of open area to bin size is small, at least 200 per cent should be added to measured air-flow requirements.

2. Where the ratio of open area to bin size is large, not less than 15 per cent should be added.

TABLE 3.—*Final Estimate of Air-flow Requirements Compared to Experimental Data*

Location of Exhaust Hood	Measured Volumes, Cu. Ft. per Min.		Final Estimate, Cu. Ft. per Min.	
	Each	Total	Each	Total
Inside surge bin......	34,500	34,500	40,000	40,000[a]
Front of ore drops...	6,400	12,800	7,700	15,400
Rear of ore drops....			1,300	2,600
Total............				58,000

[a] Increased over measured requirements to compensate for opening of chutes, doors, etc.

Table 3 shows the final estimate of air volumes to be exhausted through the various hoods of the completed installation.

Fig. 6 is a schematic diagram, drawn isometrically, of the piping layout of the exhaust ventilating system as installed late in 1939. Not shown is the dust collector, a wet scrubber developed by the Utah Copper Co., situated on the discharge side of the fans so that entrained moisture will not cause caking on the fan blades.

While the use of dampers or other restrictions in the line to secure proper distribution of air flow is widely employed, such devices are, at best, unsatisfactory. They are the first parts of an exhaust system to wear out and replacements are not always properly made. In addition, workmen not infrequently tamper with manual controls, thus disturbing the performance and efficiency of the entire installation. The use of properly adjusted pipe sizes to balance air-flow distribution makes possible a foolproof unit, either operating as originally designed or not operating at all.

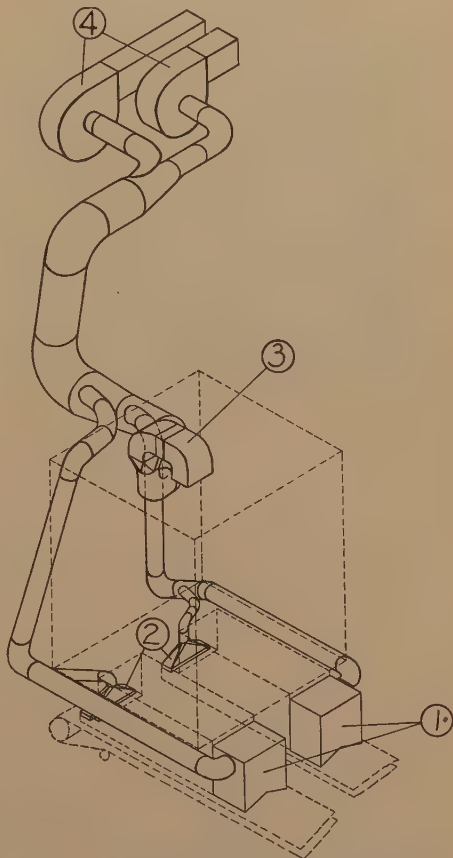

FIG. 6.—ISOMETRIC SKETCH OF PIPING LAYOUT IN EXHAUST SYSTEM OF INTERMEDIATE CRUSHING PLANT.

1. Hoods on front of ore drops.
2. Hoods on rear of ore drops.
3. Surge-bin hood.
4. Sturtevant Planovane Exhausters.

Dust collector not shown. Dotted lines show surge bin, two ore drops, two 54-inch conveyors

The method adopted for balancing exhaust ventilating systems is that advocated

* For test openings, existing doors in the sides of the bin about halfway down from the top were utilized. There were two doors on the north side containing 15 sq. ft. of open area each, and two on the south side, one containing 15 sq. ft. and the other 14.7 sq. ft. of open area.

by Drinker and Hatch[2] and others, involving the hydraulic relationship

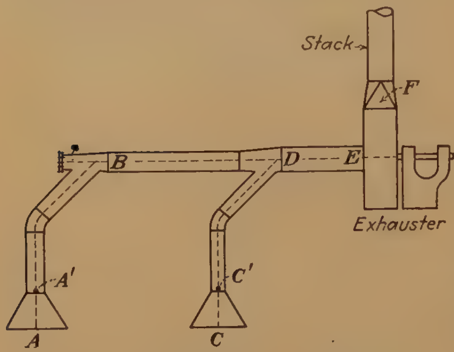

FIG. 7.—SIMPLE EXHAUST SYSTEM.

$$K_b Q_b^2 = K_m Q_m^2$$

where Q_b = rate of air flow in branch pipe,
Q_m = rate of air flow in part of system that lies upstream from branch,
K_b = pressure loss per unit of air flow through branch piping,
K_m = pressure loss per unit of air flow through part of system that lies upstream from branch.

When the simple exhaust system shown in Fig. 7 is balanced, the *total pressure* through branch *CD* will equal that through *ABD* at the desired rates of air flow.

In adjusting pipe sizes to satisfy the basic equation of flow, it was necessary to keep air velocities above the minimum required to transport the collected material. Based on Dalla Valle's[7] formulas, the following expressions gave these transporting velocities:

$$V_h = 16{,}200 \times \frac{S}{S+1} \times d^{0.4} \times \frac{W_1}{W_2}$$

and

$$V_v = 54{,}700 \times \frac{S}{S+1} \times d^{0.57} \times \frac{W_1}{W_2}$$

where V_h = required transporting velocity in horizontal pipes,
V_v = required transporting velocity in vertical pipes,
S = specific gravity of conveyed material,
d = diameter, in feet, of largest particle to be conveyed,
W_1 = air density at standard conditions,
W_2 = air density at operating conditions.

In actual practice we have found that velocities of not more than 3600 ft. per min. and not less than 3000 ft. per min. are adequate for the average siliceous dust. Lower velocities permit settling of dust in piping, while higher velocities promote abrasion and, in wet weather, sticking of certain minerals in pipe bends, etc. Frequently, however, high velocities are required in certain parts of an exhaust system in balancing short branch pipes against the rest of the system.

Several different methods are in use for determining the over-all static pressure of an exhaust ventilating system. According to the test code of the National Association of Fan Manufacturers,[8] the operating pressure equals the numerical sum of the total pressure at the fan inlet and the static pressure at the fan discharge. Many exhausters are rated according to Combined Suction-Pressure, or the numerical sum of the inlet and outlet static pressures. The latter method has been adopted by the Utah Copper Company.

As a simple example, the operating pressure of the exhaust system in Fig. 7 would be the sum of the *total pressure* (hood loss) at A' and the static losses from A' to B, B to D, and D to E minus the velocity pressure at E (numerically the sum without regard to sign) plus the static pressure at F, or, since S.P. = T.P. − V.P.,

$$\text{S.P.}_{E(-)} = \text{T.P.}_{E(-)} - \text{V.P.}_{E(+)}$$

or numerically

$$\text{S.P.}_E = \text{T.P.}_E + \text{V.P.}_E$$ and the Combined Suction-Pressure

$$\text{C.S.P.} = \text{S.P.}_E + \text{S.P.}_F$$

After requirements regarding proper transporting velocities were satisfied and pipe sizes further adjusted to ensure the proper distribution of air flow, check calculations indicated that the air distribution shown in Table 4 would take place. Where expected values varied by more than 10 per cent from required air-flow rates, the pressure drop through the part of the system in question was revised by changing pipe sizes, radii of elbows, etc. Since all piping was made in the shops of the Utah Copper Co., diameters could, if necessary, be adjusted to the nearest quarter inch. Piping was made of tank steel, ⅛ in. or more in thickness.

TABLE 4.—*Expected Air flow Rates, as Calculated by Equation of Flow Method, Compared to Required Air Flow at Each Hood*

Location of Hood	Required Air Flow, Cu. Ft. per Min.	Expected Air Flow from Calculations, Cu. Ft. per Min.
Surge bin	40,000	39,926
Front, west ore drop	7,700	7,702
Rear, west ore drop	1,300	1,311
Front, east ore drop	7,700	7,736
Rear, east ore drop	1,300	1,310
Total	58,000	57,985

The completed unit handles 58,000 cu. ft. per min. Two Sturtevant No. 90 Planovane exhausters were installed in parallel, each drawing 29,000 cu. ft. per min. against 6.78 in. of water Combined Suction-Pressure at 627 r.p.m., requiring 58 b.hp. each. These exhausters discharge into a special 10 by 10 by 30-ft. Utah Copper type wet scrubber, using 1.3 gal. of water per minute per 1000 cu. ft. per min. of air handled. This scrubber is operating in excess of 99 per cent efficiency by weight (about 92 per cent by dust count) at a 3-in. pressure drop. All fan calculations and pressure losses are based on 70° air at 4500 ft. altitude.

Unfortunately, this exhaust system was not placed in operation at the Magna plant until the beginning of the wet season, so that no representative dust counts (the best criterion of efficiency) have been made. However, an installation, based on similar

FIG. 8.—FANS AND WET SCRUBBER OF EXHAUST VENTILATING SYSTEM OF SECONDARY CRUSHING PLANTS.

principles, in the secondary crushing plant of the Arthur mill has reduced the dust concentration there to the figures given in Table 5.

TABLE 5.—*Dust Concentrations Found in Secondary Crushing Plant of Arthur Concentrator after Completion of Exhaust Installations*

Location of Hood	Number of Samples	Dust Concentration, Millions per Cubic Foot		
		High	Low	Average
Operating level	6	2.7	1.0	1.4
Crusher-discharge level	5	4.0	1.7	2.4
Conveyor tunnel	6	7.4	5.3	6.1

SUMMARY

A description is given of the methods employed at the Arthur and Magna concentrators of the Utah Copper Co. in determining air-flow requirements for the exhaust ventilation of large ore-crushing plants. Computations used in designing exhaust systems are discussed in general

terms. The ventilating installation in the secondary crushing plant of the Magna concentrator is described to illustrate the procedure followed.

The ventilation of large ore-concentration plants presents problems peculiar to that industry, partly because of the relative importance of conveying material by gravity. No method has been developed for the estimation of rates of air flow induced by falling ore in enclosed spaces. Effective enclosing will greatly reduce ventilation requirements for preventing outward leakage of dust-laden air.

The reliability of the measurement of air currents escaping from openings in bins and chutes is influenced greatly by the relative size and location of the openings.

It is seldom feasible to ventilate feeders and ore drops from within the bin. Separate exhaust hoods are required.

The compilation of specific data on existing exhaust ventilating installations would make available to ventilation engineers much valuable information. It is unfortunate that these data are frequently treated as trade secrets.

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Developments in Ball-mill Grinding Practices at New Cornelia

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(New York Meeting, February 1941)

The literature of milling is replete with papers devoted to the subject of ball milling, all of which no doubt have contributed in one way or another to progress in that art. In this paper reference will be made to only a few of the many authors, but the value of the contributions of all who have written on that subject is hereby acknowledged.

With a few exceptions the efforts of most students of the subject have been devoted to laboratory investigations of principles. Davis[1] was one of the first to report on plant-scale investigations and to present to mill men certain criteria for the guidance of ball-milling practices. His conclusions with respect to the significance of sizing analyses of classifier sands in their relation to the factors of mill speed and ball sizing are particularly referred to. Those conclusions were:

1. If the balls are large or the speed of the mill is high, crowding will appear at the finer sizes in the classifier sands.

2. If the balls are small or the speed is low, crowding will appear at the coarser sizes in the classifier sands.

3. The indications are that best efficiency is obtained when the screen analysis of the sands show a minimum crowding at any size.

The last conclusion was qualified by the statement that it had not been proved conclusively.

It is surprising, however, that the literature of later years reveals little evidence that millmen have made any serious attempt to apply those criteria. This may, perhaps, be explained by the fact that the average plant-scale ball-milling problem presents so many variables that an investigation of any one of them encounters the problem of control of the others—a problem that often is unsurmountable in a small plant. In a large plant, in which there is a duplication of grinding units in parallel operation, opportunity may be afforded for determining the value of such criteria. Nevertheless, a study of published data from grinding circuits of both large and small plants suggests that either Davis' criteria have been tried and found inapplicable or have never been given consideration.

For the purpose of testing Davis' conclusions and with improvement of ball-milling efficiency as the final objective, a test program was initiated in the concentrator of the New Cornelia branch of the Phelps Dodge Corporation early in 1938. The practices of that plant have already been described.[2] The fine-grinding units are arranged so that any one of eight units may be operated as a test unit and its results compared with those of the remaining units. All units receive similar feed from a common storage bin, in which the product of the fine-crushing plant is bedded by traveling tripper. The ball mills are all similar in size and operated at the same speeds, and are individually metered. The feeder conveyors of each unit are equipped with Merrick weightometers. The test program as initially outlined proposed investigation of the following:

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[1] References are at the end of the paper.

1. Grinding flowsheet: insofar as this could be done without major reconstruction.
2. Ball-charge rationing.
3. Low-discharge (grated-end) mills.

Up to the time of this writing the test work has been completed on only the first two items, and the results of that work form the basis of this paper.

Initial Ball-rationing Investigation

A modified two-stage grinding flowsheet had been in use for a number of years. The flowsheet of a standard mill unit is shown in Fig. 1. Included also are average tonnage, power and other data, as well as Coghill[3] diagrams of representative screen analyses of the feed, the primary and the secondary ball-mill discharges. In the application of Davis' criteria in this study, sampling accuracy dictated the use of screen analyses of ball-mill discharges rather than those of classifier sands. It is obvious, of course, that the percentages of the sizes finer than the limiting size to which grinding is being carried—minus 65 mesh in this case—have no significance. The primary mills, in which the maximum ball size was 3 in. and make-up was also of that size, readily reduced the coarser sizes in the new feed with resultant crowding of sizes near the limiting overflow size. The secondary mills, with maximum ball size of 2½ in. and make-up in the same size, treating a portion of the classifier sand derived from the primary mill product, accomplished further crowding of material in the intermediate sizes. Neither the product of the primary mills nor that of the secondaries could be considered very good classifier feed. The crowding of intermediate sizes, therefore, reflects not only poor grinding practices but also the poor classification that accompanies such practices.

The screen analyses of the mill discharges indicated that either ball-mill speeds were too high or balls were too large in both the primary and the secondary mills. Inasmuch as mills were operated at speeds approximating only 70 per cent of critical, reduction of speeds was not attractive. Some improvement in the efficiency of power application might have been obtained but most certainly at the expense of a reduction in total plant capacity. Accordingly, attention was centered on the possible benefits to be derived by changes in ball-charge sizing.

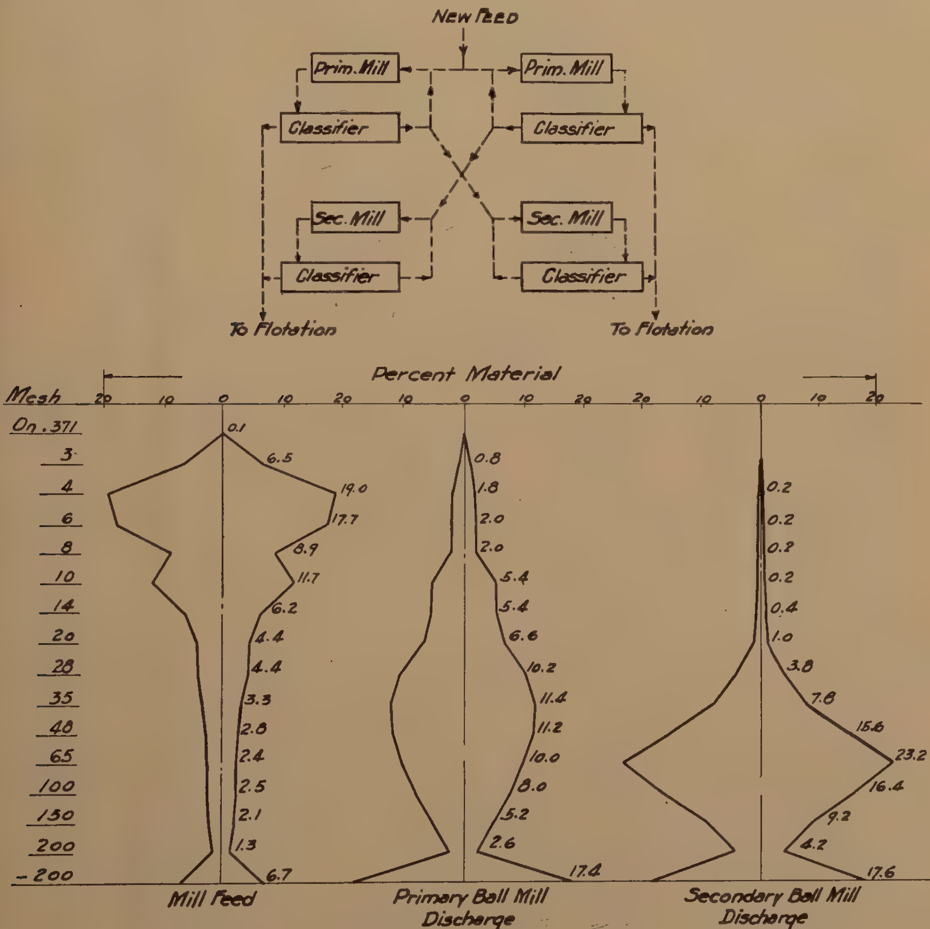
In the initial test an attempt was made to effect such changes by the simple expedient of a change in the size of the balls added as make-up. A section was chosen as a test section and the size of balls in the make-up to the primary mills of that section was reduced from 3 in. to 2½ in. The size of the balls added to the secondary mills also was reduced from 2½ in. to 2 in. This make-up practice was followed over a period of several months to insure stabilization of the respective ball charges. Tonnage and power data on the eight sections of the plant provided means for measuring the results of test-section practices and periodic sampling of products assured that plant grinding standards were being maintained. Fig. 2 presents diagrams of screen analyses typical of conditions obtaining in the test section after about three months of test procedure. Included also are comparative tonnage and power data for the test section and three adjacent standard sections covering periods both prior to and during the test.

The results of this first test were disappointing. A primary mill discharge was obtained in which there was a minimum crowding of material at any size. The secondary mill discharge also showed considerable improvement over prior experience. Notwithstanding, except during the second month of the test, power was less efficiently applied than in standard practice and there was a definite reduction in tonnage ground during the later days of the test. No means were available for accu-

rately determining tonnage finished in the respective mills, hence it seemed likely that primary mills did all that could be expected and that secondary mills were the inefficient ones. Viewed in the light of later experience it is obvious that ball charges in both primary and secondary mills were still poorly rationed for the work to be done

Flowsheet Investigations

Attention was next directed to the possibilities of changes in the grinding flowsheet. It was reasoned that if the balls were too large in standard mill charges, such charges were in effect capable of reducing larger amounts of the coarser sizes in their respective feeds. The question arose as to



Data -

Ball makeup Size - Primary Mills - 3", Secondary Mills - 2½"
Average Section Tonnage - 2460
Power per Ton - 6.91
Grind - % of Classifier Overflow on 65 Mesh - 11.1

Fig. 1.—Original two-stage flowsheet.

methods of accomplishing the desired changes in feed sizing. Insofar as primary grinding was concerned, this could be readily accomplished by passing all new such a flowsheet was easily set up and also because one of the three mills to be used as secondaries had a primary ball charge and the other two had secondary ball

Data -

	Test Section	Standard Section	Test Section % of Standards
Averages prior to tests - Tons per day	2504	2553	98.08
KW hrs. per ton*	6.62	6.51	101.69
Averages 2nd Month of Test - Tons per day	2538	2577	98.48
KW hrs. per ton*	6.28	6.40	98.13
Averages last 12 days of test - Tons per day	2404	2571	93.50
KW hrs. per ton*	6.81	6.64	102.56

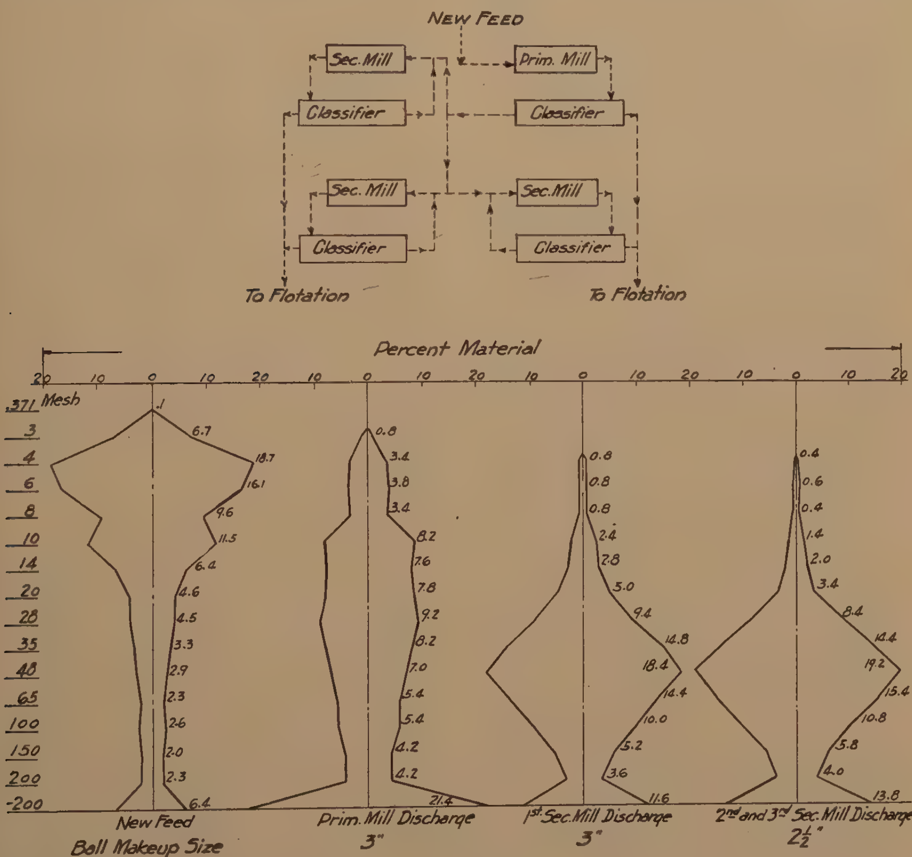
* Note - Power is Ball Mill power exclusive of Classifier power

FIG. 2.—TYPICAL CONDITIONS IN TEST SECTION AFTER ABOUT THREE MONTHS OF TEST PROCEDURE.

feed to one of the two primary mills available on a section. It was expected that the volume of new feed to this mill would make necessary open-circuit grinding. That was not objectionable if enough grinding could be accomplished so that by classification of the discharge a sand could be obtained that would be suitable feed for the other three mills operating as secondaries. The idea was particularly attractive because charges. There was thus offered an opportunity to get more data on ball rationing.

Fig. 3 presents diagrams of representative screen analyses and comparative tonnage and power data obtained during the testing of this flowsheet. The diagrams represent product sizings prevailing during the first 15 days of testing. In this period normal ball make-up practice was in vogue; that is, 3-in. balls were added to

the primary mill and to one of the secondaries and 2½-in. balls were added to the other two secondaries. The screen analysis of the discharge of the open-circuit primary sizes in the new feed when unhampered by a circulating load containing a preponderance of intermediate sizes. The discharges of the secondary mills show the usual



Data -

	Test Section	Standard Section	Test Section % of Standard Sections
Averages prior to tests - Tons per day	2430	2502	97.1
KW hrs. per ton	6.78	6.63	102.2
Averages 1st 15 days of Test - Tons per day	2550	2471	103.2
KW hrs. per ton	6.57	6.80	96.6
Averages last week of test - Tons per day	2849	2628	108.41
KW hrs. per ton	5.96	6.47	92.12

FIG. 3.—MODIFIED TWO-STAGE FLOWSHEET.

mill is of particular interest because of the absence of crowding of material at any size. It also reflects the capacity of the primary ball charge to reduce the larger crowding of sizes, though modified to some extent; probably because of the uniform sizing of feed delivered to them. An appreciable increase in section capacity with

coincident reduction in power requirement resulted from the application of this flowsheet. It was noted, however, that pool conditions in the secondary classifiers were bad, in that there was marked accumulation of critical sizes, as indicated by banking of such material at the overflow weirs. Need for further changes in all ball charges, particularly those of the secondary mills, was indicated.

Study of the primary mill discharge suggested that some improvement in grinding could be effected by the use of two sizes of ball make-up to that mill. Earlier experience had demonstrated the need for some 3-in. balls in the charge (see Fig. 2) but probably not as many as was provided in the normal primary charge. There was an apparent need for a preponderance of sizes that would be adapted to reduction of material in the size range 10 to 35 mesh, inclusive. Ball make-up practices involving addition of balls of one maximum (or minimum) size could not be expected to yield an equilibrium charge having the necessary distribution of sizes. Accordingly, make-up to the primary mill was changed; instead of balls all of 3-in. size half the amount was made up of 3-in. balls and half of 2-in. balls. At the same time, make-up to one of the secondaries was changed from the 3-in. to the 2½-in. size and to the other two secondaries from 2½-in. to 2-in. size.

The results obtained by these changes are given in the last tabulation of Fig. 3. Screen analyses are not presented because they showed only minor changes over those presented in Fig. 3. That of the primary mill discharge showed more uniform size distribution, owing to some increase in the percentages of the 4, 6 and 8-mesh sizes. Those of the secondary mill discharges showed some reduction in amounts of 35, 48 and 65-mesh sizes. The increase in tonnage ground and the reduction in power requirements resulting from the changes in ball make-up practice were gratifying. However, conditions in the circuits of secondary mills remained bad and it was evident that they could not be improved except by the use of very small balls. It seemed unlikely that sufficient gains in grinding efficiency would result to justify the higher costs incident to the application of such balls.

It is worthy of mention at this point that a three-stage grinding flowsheet was also tried. By means of it a tonnage gain of about 7 per cent over the standard sections, with a power saving of 1 per cent, was realized. In the secondary and tertiary stages of that flowsheet, the problem of properly rationing ball sizes to feed sizes was again encountered. In the tertiary stage, using ball charges with a maximum ball size of 2 in., the circuits became congested with material of 35 mesh and finer and efficient classification was impossible. This condition is believed to have limited the capacity of the three-stage unit.

Notwithstanding the improved grinding efficiency obtained by both the modified two-stage and three-stage grinding methods, neither flowsheet was considered suitable. The following criticisms were applicable:

1. Operating flexibility of the mill sections would be impaired by their use. Any loss of time on a primary mill would occasion a similar loss of time on three other mills.

2. Ball-mill maintenance was complicated because of the use of different types of ball charges.

3. The closely rationed charges required for best efficiency under average ore conditions might be ill suited to marked changes in character of feed resulting from ore changes.

4. The undesirable conditions shown to obtain in the second-stage and third-stage circuits would always present the operating problem of balancing overgrinding against high oversize.

Out of the foregoing test work, however, came certain conclusions as to the best

methods of improving grinding efficiency. First, it was demonstrated, in some measure by positive results but largely by negative results, that rationing of ball charges involved something more than a selection of the maximum (or minimum) ball size to be used as make-up; and, secondly, that given proper ball rationing, single-stage closed-circuit grinding methods offered possibilities for improving grinding efficiency without the drawbacks inherent in the more complex flowsheets. The experience with the open-circuit primary mill in the earlier tests was in large measure responsible for this latter conclusion.

Single-stage Grinding Tests

A section of the mill was converted to single-stage operation by providing for apportionment of new feed between the four mills. No immediate change was made in ball charges—two of the mills having standard primary charges with maximum ball size of 3-in. diameter and two having secondary ball charges with maximum ball size of 2½-in. diameter. It was anticipated that, owing to changes in the character of the respective classifier feeds, some improvement in efficiency over the standard sections might be shown. That this occurred is demonstrated in Table 1.

Table 1.—*Comparative Tonnage and Power Data*

Duration	Test Section	Standard Sections	Test Section, Percentage of Standard Section
For month prior to test:			
Tons per day....	2,667	2,557	104.30
Kw-hr. per ton...	7.06	6.65	106.17
Total kw-hr. per day..........	18,829	17,004	110.73
First week of test:			
Tons per day....	2,872	2,646	108.54
Kw-hr. per ton...	6.44	6.27	102.71
Total kw-hr. per day..........	18,496	16,590	111.49

The single-stage section showed a relative capacity 4.24 per cent greater than when operated as a standard section and a 3.46 per cent reduction in power per ton. It is to be noted that this section had a capacity 4.30 per cent greater than the standard sections prior to the change but also required 6.17 per cent more power per ton. These differences in capacity and power in this instance are related to the age of mill linings. The rail-type lining used is about 4½ in. thick when new and about 1½ in. thick when discarded. During the life of the lining, of course, there is a progressive increase in mill diameter, weight of ball charge and power input. Experience has shown that these changes increase grinding capacity but lessen efficient application of power. Although the power consumption per ton reflects these conditions, the total power requirements of the test section and the average standard section are presented in Table 1 as a better measure of ball-mill conditions. The grindability of the ore also is reflected in the total power figures and particularly in the requirement per ton. The gain in average daily tonnage shown for the standard sections during the first week of the test and the accompanying drop in power indicate an improvement in grindability.

In Fig. 4 are presented representative diagrams of ball-mill discharge sizings resulting from the various ball make-up practices indicated. Diagrams *A* and *B* reflect conditions prevailing with the two types of mill charges in use during the initial single-stage test. Both discharges show a crowding of the intermediate sizes and hence poor ball rationing. The tonnage gain achieved by the flowsheet change must be attributed to the improved grinding conditions resulting from the change in the character of feed delivered to the two mills that formerly had been used as secondaries.

Having in mind the results of the two-stage grinding tests, a make-up practice using 50 per cent 3-in. balls and 50 per cent 2-in. balls was applied to all mills of

the test section as an initial attempt at charge rationing. This practice led to some improvement in power efficiency,

In an attempt to minimize the testing necessary to establish the most desirable ball-charge sizing, recourse was had to the

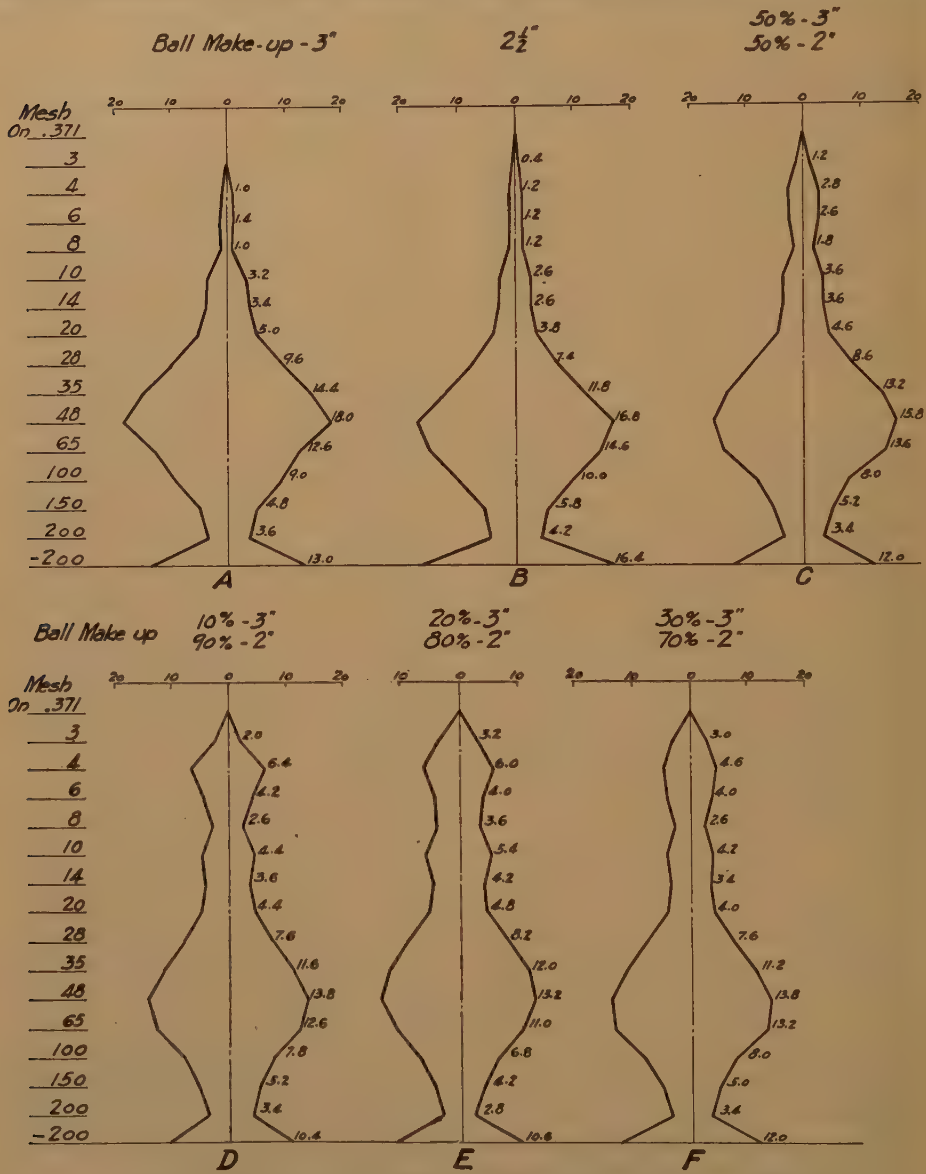


FIG. 4.—SIZING DIAGRAMS, SINGLE-STAGE GRINDING BALL-MILL DISCHARGE.

and the change in mill-discharge screen analysis shown by diagram *C*, Fig. 4. However, it was still evident that the ball charge contained too many large balls.

formula, $D^2 = Kd$, for determining optimum ball size as proposed by Coghill and DeVaney.[4] On the basis of this formula, certain assumptions as to ore character,

screen analyses of ball-mill feed and ball wear, it was decided that the equilibrium ball charge resulting from a make-up practice involving 10 per cent 3-in. balls and 90 per cent 2-in. balls would be satisfactory. It was expected that the 3-in. balls, through wear, would provide adequate amounts of the ball sizes required for the reduction of feed particles in the 6 to 10-mesh range and the 2-in. would take care of the remainder of the feed. Need for balls smaller than 2-in., preferably 1-in., was indicated but economy dictated use of the 2-in. size.

Diagram *D*, Fig. 4, of a mill discharge when treating ore of medium hardness, shows that objectives were in part attained. On harder ores it was found that the proportion of large balls in the charge was inadequate. The proportion of 3-in. balls was increased first to 20 per cent of the total and finally to 30 per cent, the proportion used in current practice. The diagrams *E* and *F* of Fig. 4 show the influence of the respective changes on the sizing of mill discharges. In Table 2 are presented tonnage and power data for the test section in comparison with similar data for the standard sections.

TABLE 2.—*Comparative Tonnage and Power Data, Averages for One Month*

Data	Test Section	Standard Sections	Test Section, Percentage of Standard Section
Tons per day.......	2,891	2,654	108.93
Kw-hr. per ton.....	6.26	6.57	95.28
Total kw-hr. per day	18,098	17,437	103.79

Comparison of Tables 1 and 2 shows that the principal gain achieved by ball rationing seems to have been a 7.43 per cent reduction in kw-hr. per ton. It is significant, however, that the average total power consumption of the standard sections increased, whereas that of the test section decreased. The increase in total power and the corresponding increase in kilowatt-hours per ton for the standard sections indicated greater average liner age and, also, a decrease in grindability of the ore treated during the period. Inasmuch as the test section maintained the tonnage advantage shown in the first week of testing as a single-stage unit, it was concluded that a definite gain in capacity had been achieved by ball rationing. That this conclusion was correct has been in large measure verified by subsequent mill experience incident to the change of all mill sections to single-stage and adoption of the ball-rationing practice developed by the test program.

Referring again to Fig. 4 and the diagram *F*, it is apparent that the optimum screen analysis was not achieved. That is attributed to the fact that, as anticipated, the use of 2-in. balls for make-up did not provide the necessary amounts of smaller balls in the charge. There can be little question that further improvement in grinding efficiency could be obtained by the use in the make-up charge of 30 to 40 per cent of balls of 1-in. size. It seems equally evident that such improvement would be evidenced by a change in screen analysis toward that uniformity indicative of such a result. The problem of determining whether use of such small balls would be economically justified remains for future investigation.

The single-stage grinding practice, although it did not yield results quite the equal of those obtained by the modified two-stage practice (Fig. 3), was adopted as the most desirable flowsheet for New Cornelia conditions. The objections to multistage flowsheets have already been given. It is not intended to imply that multistage grinding if fully developed by the use of proper ball rationing would not prove highly efficient in the application of power to grinding; it is questionable, however, whether such efficiency could be justified on an economic basis.

Equilibrium Ball Charges

Sizing analyses were made of certain of the ball charges involved in this investigation to determine results of ball wear and also the relationship of charge sizing to the sizing of composite mill feeds. In Fig. 5 are presented diagrams of composite mill feeds in the original two-stage grinding flowsheet and that of the mill feed in single-stage grinding conditions. Included, also, are sizing diagrams of a primary-mill equilibrium ball charge and the equilibrium ball charge used in single-stage grinding. The

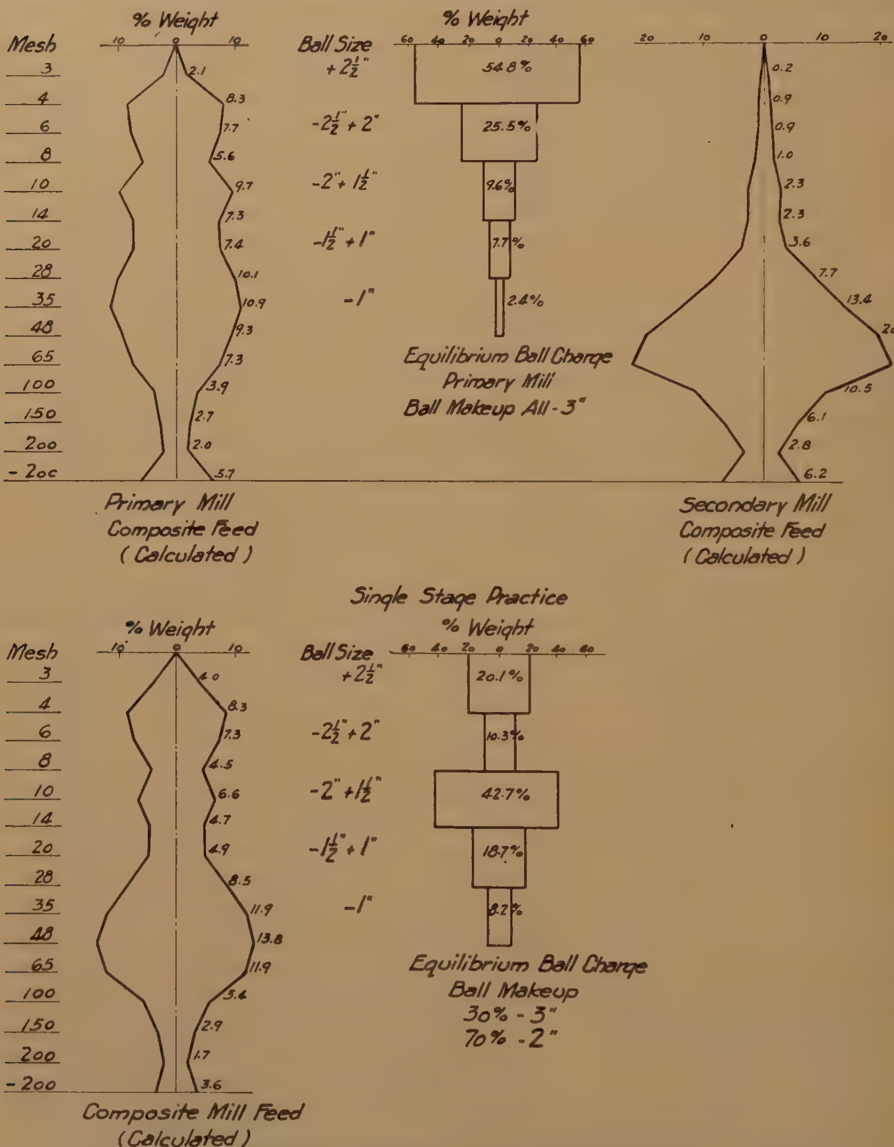


Fig. 5.—Relation of ball charge to mill feed, original two-stage practice.

uniformity of sizes in the primary mill feed is noteworthy and is explained by the fact that with the original two-stage grinding practices the ratio of circulating load to new feed was low. The primary mills each received one half of the new feed to a section and a portion of their respective classifier sands was advanced as feed to the secondary mills. Circulating load, as a result, was only about 200 per cent of the new feed.

The primary-mill equilibrium ball charge appears to have been reasonably well rationed to the composite feed. A lower percentage of plus 2½-in. balls and a greater percentage of balls 1 in. and smaller would have improved the grinding result. Comparison of the results obtained when using a similar charge in an open-circuit mill treating all new feed (Fig. 3) with the ball-mill discharge (Fig. 1) resulting from treatment of the composite feed is of interest in this connection. The diagram of the composite secondary mill feed is presented solely to show the extreme crowding of material at certain sizes, which results in a closed-circuit grinding operation where rationing of ball sizes was improperly done.

The composite mill-feed diagram for the single-stage practice is offered to present the relation of size distribution to ball-size distribution in the equilibrium single-stage ball charge. The crowding of material in the 35, 48 and 65-mesh sizes, material that originates largely in the circulating load, reflects the lack of small balls in the rationed charge. The circulating load in this instance was approximately 400 per cent of the new feed. The use of 70 per cent of 2-in. balls in charge make-up leads to the presence of a preponderance of balls in the plus 1½-in. size in the equilibrium charge. Seemingly these balls accomplish the reduction of material in the 10 to 28-mesh sizes, *F*, (Fig. 4) but are not so useful in the reduction of 35, 48 and 65-mesh material.

Classification

A discussion of a grinding investigation would not be complete without some mention of classification. In this test work there was available for closed-circuit operation one 78-in. Akins classifier for each ball mill. In all instances other than the three-stage testing, classifiers were operated with maximum sand loads. Under such conditions the ratio of circulating load to new feed was determined by the tonnage of new feed. Accordingly it varied between the limits of approximately 200 and 400 per cent. However, in all cases, the volume or tonnage of composite feed to the mills was reasonably close to the same. The relation of circulating load to the sizing of the composite feed in the primary mills of the original two-stage flowsheet and to that of the single-stage circuit has already been mentioned. Significant is the fact that improvement in the uniformity of mill-discharge sizing (classifier-feed sizing) effected improvement in the efficiency of classification. This is demonstrated by the percentage of finished material (minus 65 mesh) shown in the diagrams of the extremes of composite feed sizing as represented by the secondary-mill composite feed and the composite mill feed of single-stage practice shown in Fig. 5. These results enforce the conclusion that the percentage of circulating load had less bearing on the final grinding result than did proper ball-charge rationing.

Summary and Conclusions*

A series of plant-scale ball-mill grinding tests has been carried out at New Cornelia, with improvement of grinding efficiency as the objective. It was further proposed in the

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* It has been suggested by a reader of the manuscript that readers of this paper by Messrs. Barker and Lewis may find it useful to refer to the paper on Wear and Size Distribution of Grinding Balls, by F. C. Bond, which appears on pages 373 to 384 of this volume.

test program to ascertain the value of certain criteria advanced by Davis[1] for the guidance of ball-milling practices. In this work grinding efficiency was measured in relative rather than absolute terms. The conclusions drawn from this test program are as follows:

Davis' criteria have a definite value for the guidance of grinding practices, particularly in the application of ball-charge rationing.

The flowsheet of single-stage closed-circuit grinding was the most desirable for conditions at New Cornelia.

Ball-charge rationing was essential to the development of maximum grinding efficiency irrespective of the grinding flowsheet used. Its simplest application was found in single-stage grinding practices.

Ball-charge rationing does not consist of an application of the maximum (or minimum) ball make-up size, but, rather, the use of those several make-up sizes that will yield an equilibrium ball charge adapted to the reduction of the composite ball-mill feed. There was considerable evidence that for closed-circuit grinding the rationed ball charge must contain a preponderance of the sizes suited to the reduction of particles in the size range where crowding results incident to classification effects.

Effective ball rationing leads to improvement in the efficiency of classification, with its resultant effects on grinding efficiency.

The relation of cost of grinding media to total mill economy may determine whether it is practicable to apply ball-rationing practices that would yield optimum grinding efficiency.

Acknowledgment

The authors extend their thanks to the Management of Phelps Dodge Corporation for the permission to publish the results of this ball-milling investigation.

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2. L. M. Barker: Milling Methods at New Cornelia. *Trans.* A.I.M.E. (1939) **134**, 434–452.
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Crushing and Grinding Practice, Tennessee Copper Company

By J. F. Myers,* Member, and F. M. Lewis,* Junior Member A.I.M.E.

(New York Meeting, February 1940)

The Tennessee Copper Company's operations are in the Ducktown Basin, in the extreme southeast corner of Tennessee. The ore is of the heavy sulphide type, the predominating sulphides being pyrite, pyrrhotite, chalcopyrite and sphalerite. The gangue is a micaceous schist, with varying amounts of free quartz. The mining operation develops two classes of ore, each of which is concentrated by the flotation process in separate mills. Table 1 shows a typical analysis of the ore treated in each mill.

Table 1.—*Typical Analysis of Ore Treated*

Mill	Tons Capacity	Composition, Per Cent			
		Cu	Fe	S	Zn
London	1,350	1.41	28.4	20.2	1.10
Isabella	850	0.69	44.7	31.6	0.31

London Mill

The London mill was built in 1922, with a capacity of 400 tons per day. This was gradually increased to 600 tons by 1926, to 900 tons in 1928, and to 1350 tons in 1938. Fig. 1 shows the crushing and grinding flowsheet in 1928, when the capacity was 900 tons; Fig. 2, the crushing and grinding flowsheet that was developed by 1938, for the treatment of 1350 tons per day. A comparison of these flowsheets reveals that this increase in tonnage, with a subsequent improved operation, was accomplished with identically the same major equipment.

Improving a concentrator's capacity, metallurgical results, and costs as time goes by is a commonplace practice, and is accomplished in various ways, depending upon the local conditions. The reason for presenting the data of how this was accomplished in the mills of the Tennessee Copper Co. is that a fast-running rod mill plays such an interesting role

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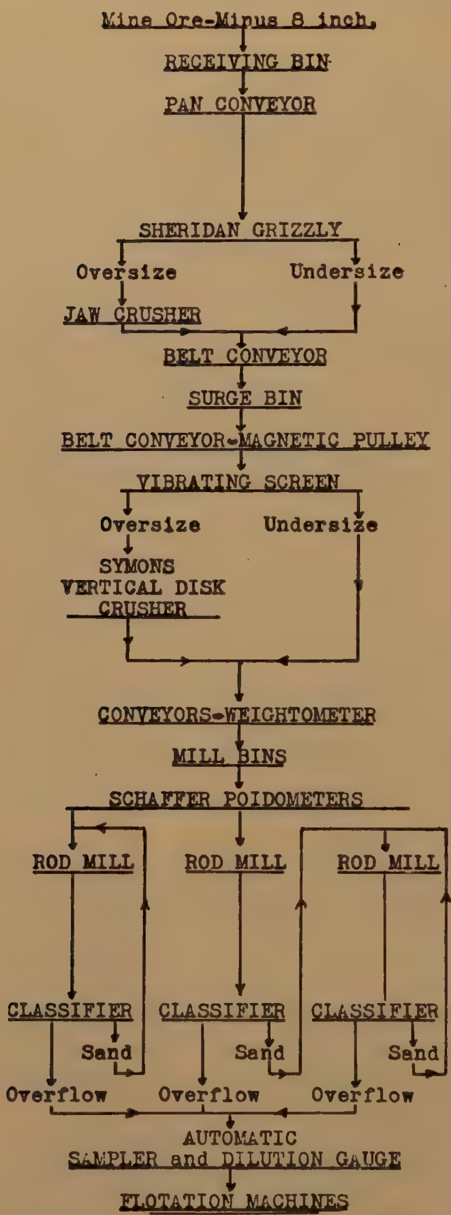

Fig. 1.

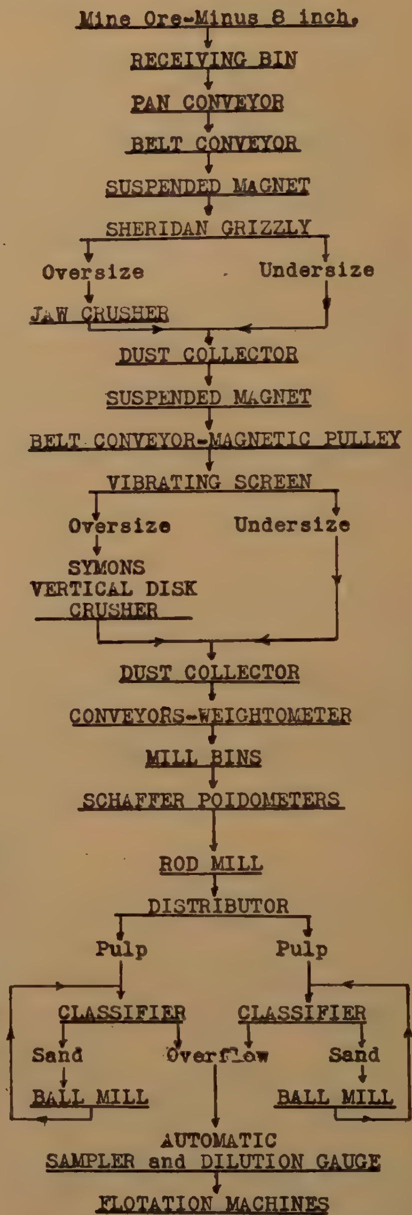

Fig. 2.

Fig. 1.—London crushing and grinding flowsheet, 1928.
Fig. 2.—London crushing and grinding flowsheet, 1938.

in the procedure. F. J. Tuck[1] and E. H. Rose[2] have contributed some interesting data in this connection. The point should be emphasized that the data presented herewith are to show the relationship held by a fast-running rod mill between the crushing plant and the fine-grinding operation. This comment is made because it will be obvious to those skilled in the art that the optimum results have not yet been reached in the secondary grinding units. The application of faster speed, smaller balls and improved classification has not been made for purely metallurgical reasons in the flotation machines.

TABLE 2.—*Data on London Crushing Plant*

	Flowsheet Fig. 1	Flowsheet Fig. 2
Primary crusher, Allis-Chalmers:		
Type	Jaw	Jaw
Size, in.	24 × 36	24 × 36
Strokes per minute	195	245
Closed setting, in.	3	2⅜
Stroke, in.	¾	⅝
Finishing crusher, Symons:		
Type	Vertical disk	Vertical disk
Size, in.	48	48
Speed of drive shaft, r.p.m.	380	415
Closed setting, in.	⅛	¼
Crusher-plant product:		
Tons per hour	121	183
Screen analysis:		
+1 in.	0.6	3.5
−1 in. + ¾ in.	3.7	5.9
−¾ in. + ½ in.	7.2	11.4
−½ in. + ¼ in.	27.8	20.7
−¼ in. + 200 mesh	53.5	51.7
−200 mesh	7.2	6.8
	100.0	100.0
Kilowatt-hours per ton, receiving bins to mill bins	1.36	1.10
Cost per ton, receiving bins to mill bins (cost covers all operating and maintenance)		$0.031

Tables 2 and 3 show a comparison of the main operating data of the London crushing and grinding section, respectively.

In Table 3 the data tabulated under flowsheet Fig. 2 were selected to show what the performance would be at 1250 tons per day, as this figure comes closest to the normal conditions. Rates of feed at 1300 and 1350 are common when low-grade ore (25 per cent Fe) is encountered. Likewise on high-grade ore (31 per cent Fe) the feed is reduced to 1100 or

[1] F. J. Tuck: Rod-mill Practice at Ray Mines Division, Kennecott Copper Corpratoion. *Trans.* A.I.M.E. (1939) **134**, 327.

[2] E. H. Rose: *Canadian Min. Jnl.* (Nov. 1937).

1150 tons per day. The maximum capacity under the present arrangement is 1350 tons per day. The quantity of the ore treated is governed by the requirements of iron concentrate, 450 tons per day, and hence the grade of the ore determines the rate of feed through the grinding section.

Within the limits of a short paper, it is impossible to present all of the data that influenced the present arrangement, but there follows a brief description of the main events leading up to the present practice.

TABLE 3.—*Data on London Grinding Section*

	Flowsheet Fig. 1	Flowsheet Fig. 2
Allis-Chalmers 6 by 12-ft. mills		
Stages of grinding	One	Two
Charged with 3-in. rods	3	1
Charged with 2-in. balls		2
Rod mill: r.p.m.	17	22
Peripheral speed (inside diameter)	298	386
Ball mill: r.p.m.		22
Peripheral speed (inside diameter)		386
Total manganese liner wear per ton, lb.	0.201	0.113
Rod consumption per ton of ore	2.736	0.695
Ball consumption per ton of ore		1.161
Dorr classifiers, 8-ft. model D		
Strokes per minute with ball mills		23.5
Strokes per minute with rod mills	26.2	
Dilution of overflow, per cent solids	30.4	32.0
Grinding-section product		
Tons per day of mill feed	900	1,250
Screen analysis:		
+65 mesh	6.0	4.0
−65 + 200 mesh	39.2	36.3
−200 mesh	54.8	59.7
Kilowatt-hours per ton, mill bins to flotation machines	10.41	8.50
Cost per ton, mill bins to flotation machines		$0.176
Kilowatt-hours per ton for crushing and grinding	11.77	9.60
Cost per ton for crushing and grinding		$0.207

It is a well-known fact that the most economical results in crushing and grinding are obtained when they are considered together. The reduction of ore particles larger than 1 in. is definitely a crusher problem. The reduction of ore particles smaller than ¼ in. is definitely a grinding matter. The reduction of ore particles between 1 in. and ¼ in. has long been a subject of controversy, and probably differs on different ores and conditions. Between the years 1928 and 1938, many different grinding flowsheets were tried in the London mill. The most outstanding fact in all of this work is the marked ability of a rod mill to reduce crushed ore from 1 in. or ¾ in. down to a 10-mesh or 20-mesh product. It became evident that "flats" or "slabs," with one dimension correct for all prac-

tical purposes, grind just as well as screened feed when ore particles have approximately equal dimensions in all directions. The test work shows that under similar conditions this is not true of a ball mill, as the circulating loads in the classifier become too large to handle unless special arrangements are made or new equipment purchased.

Obviously, if a rod mill could be made to do more work, it would greatly simplify the London crushing problem. On several occasions, rolls and other fine-crushing equipment were seriously considered, to bring the minus 1-in. crusher product down to a suitable ball-mill feed of about ¼ in., and to abandon the rod mill entirely. In the old crusher plant, this would have required considerable capital expense, because of the installation of closed-circuit screening, return belts, dust collection and other apparatus. This would have been the conventional step to take; but, in the meantime, one rod mill had been speeded up from 17 to 19

TABLE 4.—*Résumé of Grinding-section Results*

Grinding Arrangement	Rod Mill		Finished Product		
	R.p.m.	Ore Treated, Tons per Day	Kw.-hr. per Ton Ore	Plus 65 Mesh	Minus 200 Mesh
Single-stage rod mills	17	900	10.41	6.0	54.8
Single-stage rod mills	19	990	9.90	5.8	58.3
Full two-stage (16 months)	19	1242	8.33	5.1	57.0
Full two-stage (19 months)	22	1251	8.50	4.0	58.5
High-tonnage periods "A"	22	1301	.....	5.2	58.0
High-tonnage periods "B"	22	1350	.....	5.7	57.2

Note: Power includes grinding mills, classifiers pump, feeder belts and overhead crane.

r.p.m., in order to determine whether or not something could be done to take advantage of its ability to handle coarse, one-dimension slabs or flats. The results from this test were so favorable that early in 1929 the other two mills were speeded up to 19 r.p.m. During the latter part of 1929 the rod charge from one mill was removed and the mill filled with 2-in. balls. This converted mill was then put in series with one rod mill, and the principles of two-stage grinding were studied in so far as the equipment would permit.

Late in 1930 the rods were removed from the third mill, its speed was increased to 22 r.p.m. and it was filled with a ball charge graduated up to 3½ in. This mill was tried in single-stage work on the one-dimension feed from the crusher plant. The results from this test conclusively proved that the ball mill could not handle the one-dimension crusher feed, as the classifier sands were prohibitive. The only way that this mill could be made to do anything was to set the crushers for a minus ½-in. product. This test was soon abandoned.

The rod mill and one ball mill that were working in a semi-two-stage arrangement were doing so much better proportionately than the one single-stage mill that in the fall of 1934 the third mill was charged with 2-in. balls and also put in series with the rod mill as shown in Fig. 2. This change to full two-stage grinding was a major improvement in itself, but also important is the fact that it put the rod mill in the first grinding stage, where it was best fitted to treat the coarse-crusher product. The scheme worked very well except that there was a little tendency toward

TABLE 5.—*Rod-mill Operation at* 19 *and* 22 *Revolutions per Minute*

	19 R.p.m.	22 R.p.m
Rate of feed for test periods	1,254	1,244
Kilowatt-hours input to rod mill (liners about half worn)	131	136
Pounds per ton manganese ship lap liners	0.088	0.069
Life of liners, days	217	293
Pounds per ton of 3-in. high-carbon rods	0.687	0.675
Pounds of rods per operating day	861	840
Rod-mill feed:		
+½ in.	17.8	20.8
−½ in. + ¼ in.	14.4	20.7
−¼ in. + 200 mesh	60.2	51.7
−200 mesh	7.6	6.8
	100.0	100.0
Rod-mill discharge, mesh:		
+20	1.0	1.1
−20 + 28	4.8	4.9
−28 + 35	9.2	8.9
−35 + 48	12.7	12.5
−48 + 65	15.9	14.7
−65 + 100	12.4	13.5
−100 + 150	10.9	10.8
−150 + 200	7.9	7.8
−200	25.2	25.8
	100.0	100.0

overloading of the light model D classifiers in the secondary circuit when the crusher product was running a little on the coarse side, as when the mill bins were getting low or when crusher product was a little on the coarse side.

To relieve the classifiers a little and to help the crushing plant further, the rod mill was again speeded up to 22 r.p.m. early in 1936. This faster speed and more suitable liners in the secondary ball mills, which increased the power input, brought the capacity up to its present status of 1350 tons.

The effect of the two-stage arrangement is interesting, and a résumé of the results from 1928 to 1938 is shown in Table 4.

In 1939 a drive pinion was available to increase the speed to 23 r.p.m., therefore this was done. At present, there is no doubt that still faster speeds can be used.

The test work on the various rod-mill speeds was carried out between the years 1934 and 1938. Records at 19 and 22 r.p.m. are shown in Table 5. For a short period of time the speed was dropped back to 17 r.p.m., but this caused too much sand load for the secondary ball-mill classifiers, and the test run was abandoned.

TABLE 6.—*Rod-mill Action on Different Sizes of Feed, 22 Revolutions per Minute*

	Coarse Feed	Fine Feed
Tons per day to rod mill	1,200	1,200
Dilution of rod mill pulp, per cent	66.8	67.0
Crushed ore to rod mill:		
+1 in.	8.6	0.0
−1 in. + ¾ in.	16.2	1.8
−¾ in. + ½ in.	17.6	12.4
−½ in. + ¼ in.	17.4	20.3
−¼ in. + 20 mesh	20.2	33.3
−20 + 200 mesh	14.4	24.3
−200 mesh	5.6	7.9
	100.0	100.0
Rod-mill discharge, mesh:		
+20	3.3	1.4
−20 + 28	4.7	3.4
−28 + 35	9.8	6.7
−35 + 48	13.2	12.0
−48 + 65	15.3	14.1
−65 + 100	12.9	14.1
−100 + 150	10.4	11.8
−150 + 200	8.2	7.8
−200	22.2	28.7
	100.0	100.0

Normally, the London crusher plant operates one shift each day, as this provides enough crushed ore to keep the grinding section running 24 hr. The crushed ore is stored in two cylindrical steel bins 20 ft. in diameter and 40 ft. high. Naturally, the last ore out of the bins gets pretty coarse and the first out is the finest. How admirably the fast-running rod mill handles this irregularity is shown in Table 6.

The flowsheet in Fig. 2 shows that the rod mill operates in open circuit and that no classifier is used. An 8-ft. model D classifier is available to close the circuit if desired. It is entirely possible that with more suitable equipment a primary closed-circuit operation might prove of further advantage. However, it has been determined that for the tonnage being

handled and the final grind required the open-circuit mill produces a very satisfactory secondary ball-mill feed and one on which good secondary results are obtained.

Operation of Grinding Section

The ore is fed and weighed from the mill bins by two Schaffer poidometers. These are arranged with electric contacts to blow Klaxon horns if the feed hangs up, which occasionally happens when the ore is wet or when some foreign substance comes through the feeder. The rate of feed is checked once or twice a day by weighing a given length of ore on the poidometer belts.

Grinding rods are added as needed, twice each week to keep the rod load up to the correct level, which normally is about 3 in. below the center line of the mill. The plant shuts down for repairs one shift each week and at that time all the small rods are removed from the mill. A small rod is considered to be one of 1-in. dia. or less. Some few rods occasionally escape removal, break up into pieces and gradually work out of the mill. It has been found repeatedly that the mill efficiency is greatly improved if all rods less than 1 in. are removed. The scrap rods are included in pounds per ton of rods consumed, shown in this report. The scrap weight of liners is also included in the liner wear.

A great many kinds of liners have been tried. The London experience indicates that with fast-running rod mills there is little advantage in the ship-lap type over the wave type of liner. Wave liners with waves from ½ to 4 in. were tried. The slower the rotation of the mill, the more liner wave is required to keep the load from slipping. The end liners are solid one-piece castings, as they protect the ends of the mills from pulp wear and there is no possibility that pieces of rods may get into cracks, as they do in sectional liners, and thus tangle the load of rods. The shell liners are backed with ¼-in. rubber sheets 4 by 12 ft. This has been found to be far better than wood; since it gives the maximum inside mill diameter and hence the greatest capacity.

The liner bolts are made from a special steel of very high tensile strength, to permit a thin bolt head and prevent elongation. One nut properly tightened is ample to hold the bolt. The bolt holes in the shell must be true and a snug fit with the bolt must be assured.

The rod mill operates at a dilution of 65 to 67 per cent solids. The specific gravity of the ore is 3.3, which at this dilution is a relatively thin pulp. The rod-mill discharge is pumped to a distributor, which divides the pulp into two equal parts, each feeding by gravity to the two 8-ft. model D classifiers, which are in closed circuit with the secondary ball mills. The overflows from the two classifiers join together before going to the flotation machines. The classifier sands are fed to the ball-mill scoop boxes by means of a 9-in. screw conveyor, which is driven off the drive shaft of the classifier by means of chain and sprockets. The screw-

conveyor box is deep enough so that the screw always rides on ore and never wears out the box. The screw is supported at each end by plain cast-iron bearings, to which part of the mill dilution water is added for bearing lubrication. The screw conveyor is 16 ft. long and, since no bearings are used except at the ends, the sag at the center is taken care of by the screw riding on its own bed of ore. The screw lasts about 9 months and the bearings about 4 months.

The two ball mills are 6 by 12-ft. rod mills converted to a ball charge, as mentioned previously. The shell liners are the same as described for the rod mill above, the only difference being in the end liners. These likewise are cast solid and are of the "Pinguico" type. This liner, with the help of a reversed spiral screw in the trunnion liner, permits carrying the ball load to within 2 in. of the center line of the mill.

The ball mills rotate against the slope of the ship-lap liners and the rod mill with the slope of the liners. The total wear of the shell liner reported in Table 3 is 0.113 lb. per ton. This is divided between the various mills as follows: No. 1 rod mill, 0.069; No. 2 ball mill, 0.022; No. 3 ball mill, 0.022; total, 0.113.

The ball-storage bins are on a floor higher than the mill floor, and from these bins the addition of balls to the mills is by means of gravity through a pipe into the discharge end of the mills, each day. Fig. 2 shows that the two ball mills operate under identical conditions, which makes it easy to test out various kinds of balls against each other.

The ball mills operate at a dilution of 70 to 71 per cent solids in the summer months and from 66 to 69 per cent solids in the winter time. With a heavy specific gravity of ore, these dilutions are pretty thin, but it is all that can be carried without materially affecting the metallurgical results; and, after all, the purpose of the operation is to obtain the best metallurgical results. The chemical and physical reactions that go on in a grinding mill, particularly on a heavy sulphide ore, must be a very intricate affair. At the London mill it is one of the main control points of the flotation process.

All the dilution water added to the grinding section comes from a constant-level water tank, in order that a constant head may be maintained at all times on the water lines feeding the dilution water. All water valves to the classifiers and the mills are always operated wide open, and any adjustment of dilution water is made by changing the fixed orifice of the water line as it leaves the valves. This procedure may sound a little difficult, but it is not, because the feed from the crusher is uniform. The ore is weighed into the rod mill and with the dilution water at a uniform pressure there are no adjustments to make. In fact, no operator is stationed on the grinding floor.

At the junction box of the two classifier overflows, an automatic sampler cuts the stream every 15 min. At this point, there is also an automatic dilution gauge, which constantly records the classifier overflow

dilution on a chart. The gauge is sensitive and records any and all irregularities in the mills and classifiers. This gives the shift foreman ample warning to search out any rarely occurring trouble. It is almost a foolproof arrangement and produces a uniform product to the flotation machines.

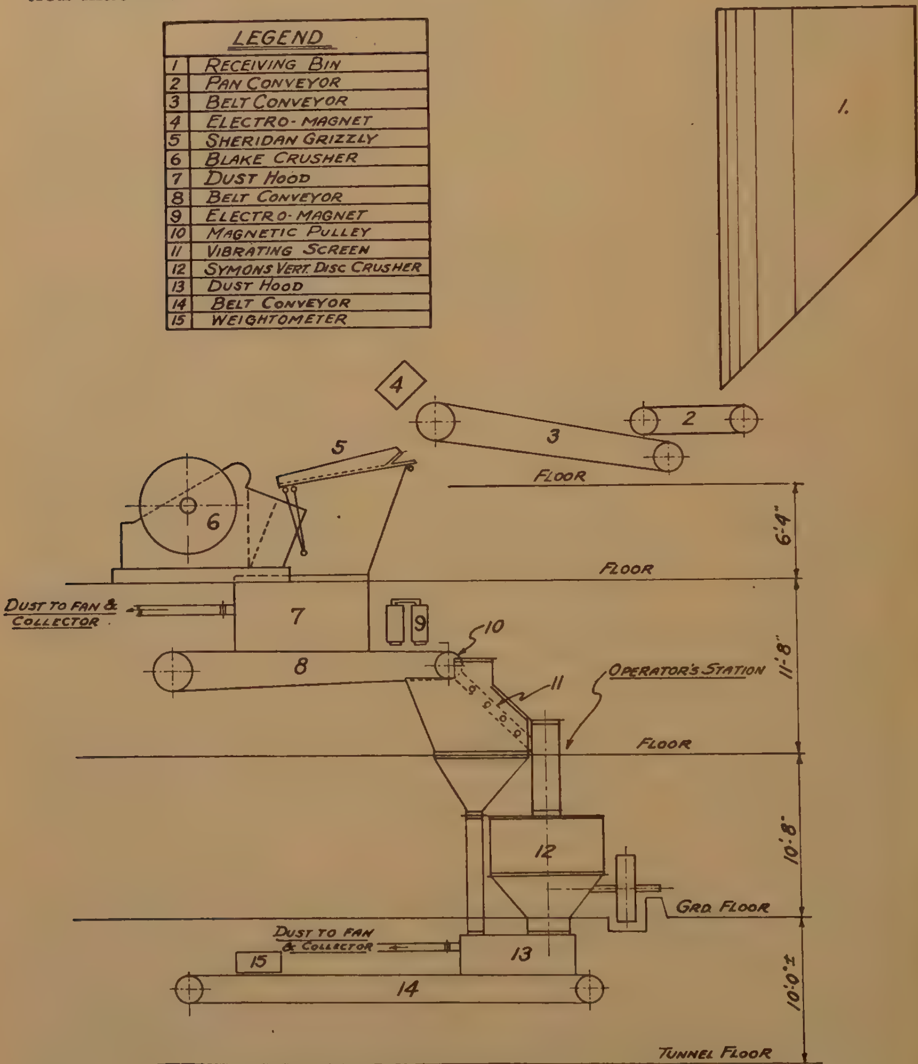

FIG. 3.—LONDON CRUSHING PLANT.

CRUSHER-PLANT OPERATION

The arrangement of a crusher plant to produce one-dimension particles for the grinding section becomes a very simple matter. In the London mill, a product having one dimension ¾ in. is correct for the fast-running rod mill in the grinding section. Fig. 3 shows an elevation of the London

crushing equipment, which requires one operator. This operator is stationed at the feed chute of the Symons crusher and regulates the quantity of feed to the jaw crusher by a pull cord attached to the ratchet

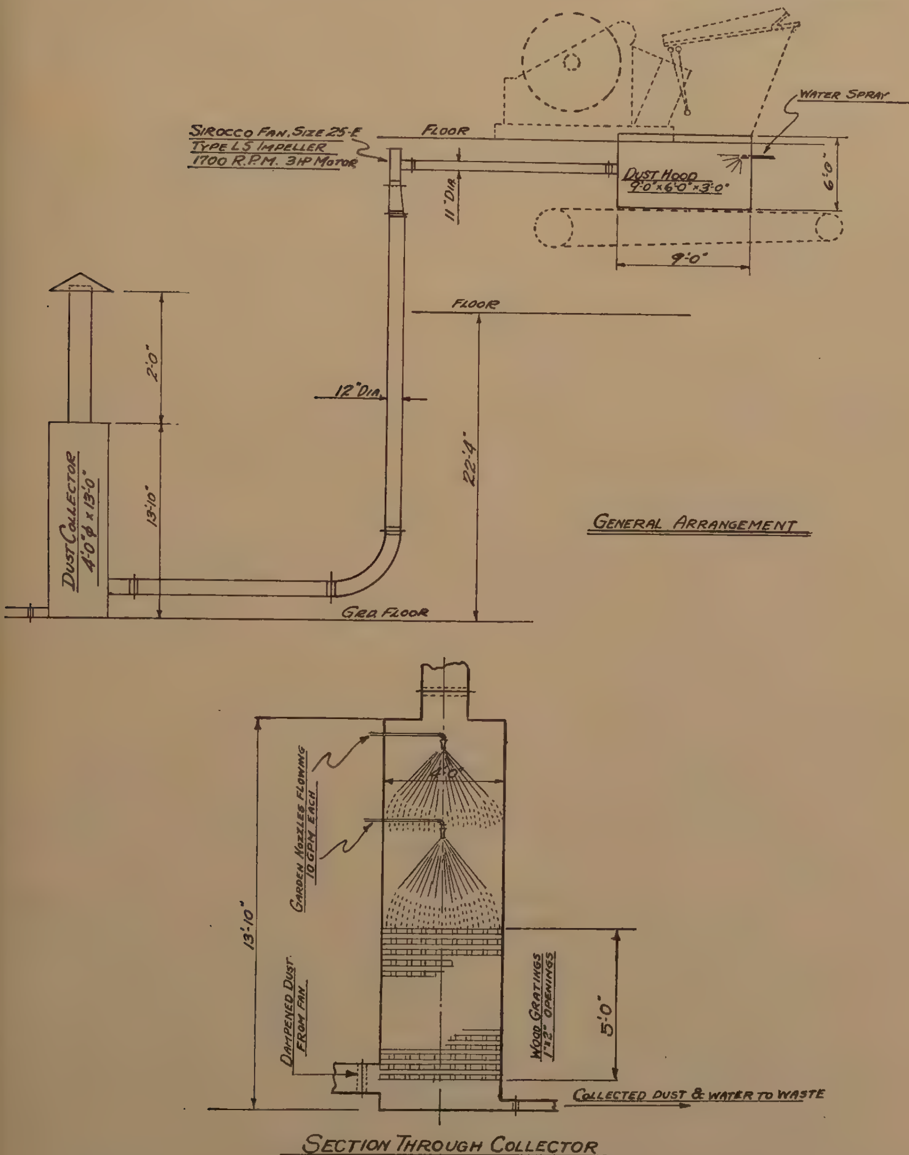

FIG. 4.—LONDON DUST COLLECTOR.

of the pan feeder on the floor above. The operator can tell by the sound of the jaw crusher just what it is doing, although normally the two crushers are synchronized so that one crushes at about the same rate as the other.

The control of the crushed ore is governed by the opening of the Symons crusher disks, of course. Each shift a sample of the crushed ore is taken and a screen analysis made therefrom. From this analysis the operator knows when to close the Symons crusher to compensate for wear. Usually he does this every other shift, but sometimes three shifts can be worked before an adjustment is needed. This gives a fairly uniform feed to the grinding mills.

Flashing telltale lights at the operator's station keep him informed that all the conveyors are running, since he has no opportunity to investigate personally. There are also light circuits to inform him when the mill bins are full; the damp ore touches a rod at the top of the mill bin and completes the electrical circuit with the steel bin, which lights the bulbs at the station.

Because of such a simple crushing arrangement, collection of dust is required only under each of the two crushers. The dust hoods under each crusher are drafted by separate fans and the fans are run at sufficient speed to ensure a draft through the ore in the crushers. By this means the dust count is held well under the figure recommended for this ore. Periodic dust counts are made by the safety department. Fig. 4 shows a detailed arrangement of the dust-collecting system. This would not be suitable for a very dry, siliceous ore, but for an ore containing 1.5 to 2.5 per cent moisture the system suffices. The collected dust and water can be saved or wasted, as desired. At London it is turned to waste.

The ability of the fast-running rod mill to grind one-dimension particles from the crusher plant produces the following advantages in the crusher plant:

1. It removes the necessity of closed-circuit crushing, with its screens and return equipment, such as conveyors or elevators. This reduces both capital and maintenance and operating expenditures.

2. It reduces the cost of dust collection, as the closed-circuit systems greatly increase the number of points that generate dust.

3. On an ore that contains some wet fines or sticky material, the closed-circuit principle of crushing is troublesome. The open circuit, as described herein, is much easier to handle.

4. In old plants, it is often possible to increase production without the installation of new equipment.

5. When ore is slightly damp, it is easier to feed the grinding mill if some coarser particles are present in the feed.

The only disadvantage, or perhaps it would be best to say the one thing to guard against, is that adequate magnet protection must be provided to take care of tramp steel. In Fig. 3, a magnet is shown guarding each crusher. This is important. The tension springs on the Symons crusher are kept a little tighter than normally is considered good practice, in order to control the crusher product; therefore it does not pass scrap steel readily. However, the use of magnets is standard

practice and no crusher is ever benefited by tramp steel, even when equipped with mechanism to pass it by.

Isabella Mill

The Tennessee Copper Co. obtained control of the Isabella mines and surface plants in the fall of 1936. The Isabella mill at that time was

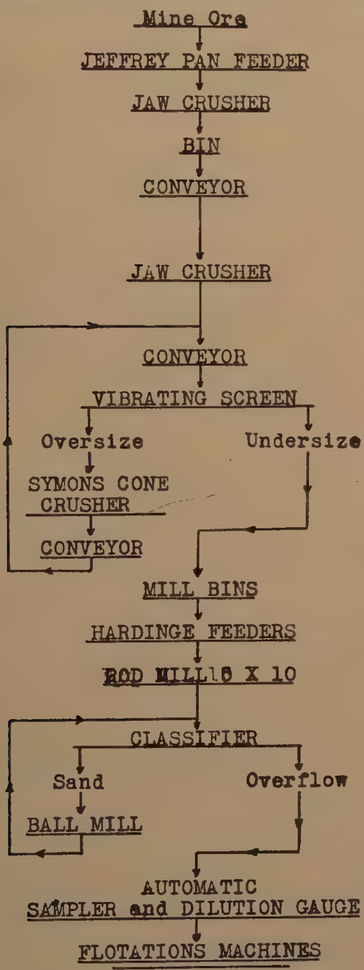

Fig. 5.

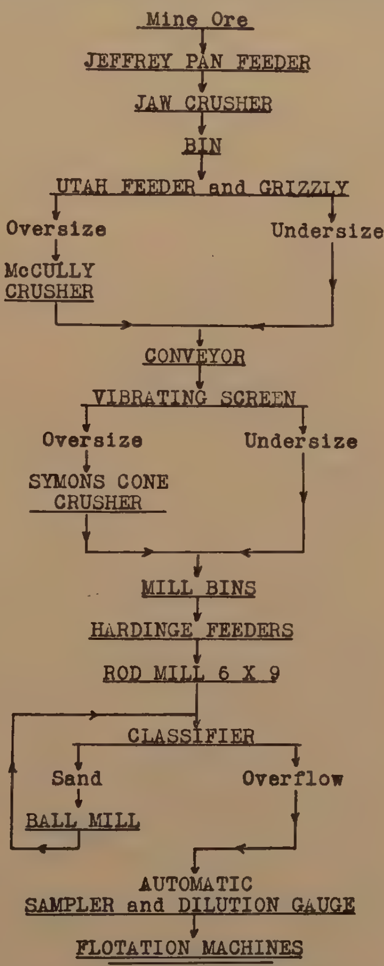

Fig. 6.

Fig. 5.—Isabella crushing and grinding flowsheet, closed-circuit crushing.
Fig. 6.—Isabella crushing and grinding flowsheet, open-circuit crushing.

treating 725 tons of ore per day. The first step in reducing the costs was to convert the single-stage rod mills into a two-stage grinding circuit, using rods in the first stage and balls in the second stage (an arrangement similar to the London flowsheet). By this means, two 5 by 10-ft. mills were able to do the work of three 5 by 10-ft. mills in single stage charged with rods. At this time the crushing plant had not been altered and the

conventional closed-circuit principle was still in use. Fig. 5 shows the flowsheet of the closed-circuit crushing plant and the remodeled grinding section. The small (5-ft.) diameter of the rod mill prevented the application of the open-circuit crushing principle established at London, because the cascade of the weight of rods contained in the mill are not enough to reduce minus 1-in. one-dimension particles.

TABLE 7.—*Data on Isabella Crushing Plant*

	Flowsheet Fig. 5	Flowsheet Fig. 6
Primary crusher:	Buchanan	Buchanan
Type	Jaw	Jaw
Size, in	30 × 42	30 × 42
Strokes per minute	195	195
Closed setting, in	3½	3½
Stroke, in	⅝	⅝
Intermediate crusher:	Buchanan	McCully
Type	Jaw	Gyratory
Size, in	12 × 24	6
Strokes per minute	290	348
Closed setting, in	1½	1
Stroke, in	¾	⅝
Finishing crusher:	Symons	Symons
Type	Cone	Cone
Size, ft	3	3
Strokes per minute	324	324
Closed setting, in	⅜	⅝
Stroke, in	1¾	1¾
Crusher-plant product:		
Tons per hour	76	161
Screen analysis:		
+ 1 in		8.3
−1 in. + ¾ in		21.8
−¾ in. + ½ in	0.6	25.6
−½ in. + ¼ in	17.9	14.3
−¼ in. + 200 mesh	72.8	25.8
− 200 mesh	8.7	4.2
	100.0	100.0
Kilowatt-hours per ton, receiving bin to mill bins	1.51	0.90
Cost per ton, receiving bin to mill bins (cost covers all operating and maintenance)	$0.079	$0.049

In 1938, it became necessary to increase the tonnage from 725 to 825 tons per day. A thorough study of the conventional application of fine-reduction crushers was made, in order to be able to crush finer, and thus pick up the required additional tonnage, without changing the grinding mills. The data obtained were compared with those of the London plant, using a fast-running rod mill on a minus 1-in. one-dimension crusher product. It was decided to follow the London practice.

Fig. 6 shows the flowsheet of the crusher plant changed from closed-circuit to open-circuit crushing, and the new large rod mill in the grinding section.

The operating data for the flowsheets of Figs. 5 and 6 are given in Tables 7 and 8. These data clearly show the manner in which the fast-running rod mill relieves the load on the crusher plant and increases the capacity of the grinding section.

TABLE 8.—*Data on Isabella Grinding Section*

	Flowsheet Fig. 5	Flowsheet Fig. 6
Primary rod mill, Allis-Chalmers:		
Size of mill, ft	5 × 10	6 × 9
Size of rods, in	3	3
Speed, r.p.m	27.2	24.4
Peripheral (inside diameter)	392	428
Rod consumption per ton of ore	0.647	0.697
Secondary ball mill, Allis-Chalmers:		
Size of mill, ft	5 × 10	5 × 10
Size of balls, in	2	2
Speed, r.p.m	30.8	30.8
Peripheral (inside diameter)	443	443
Ball consumption per ton of ore	0.632	0.632
Classifier, model D, width, ft	6	6
Classifier strokes per minute	32	32
Grinding product:		
Tons per day	725	825
Screen analysis, mesh:		
+ 65	7.8	6.0
+ 200	40.2	39.2
− 200	52.0	54.8
	100.0	100.0
Kilowatt-hours per ton of ore	5.13	6.12
Wear on manganese liners per ton of ore, lb	0.108	0.118
Operating and maintenance cost per ton	\$0.145	\$0.136
Kilowatt-hours per ton, crushing and grinding	6.64	7.02
Cost per ton for crushing and grinding	\$0.224	\$0.185

Conclusions

From the experience and data reported, the conclusion can be drawn that a fast-running rod mill deserves consideration as a fine-crushing machine on ore particles with one-dimension 1 in. or smaller because:

1. It permits open-circuit crushing ahead of it, which has many advantages.
2. It prepares an ideal feed for the ball mills that follow it.
3. The combined cost per ton for crushing and grinding is lowered.

DISCUSSION

(*E. W. Engelmann presiding*)

W. L. Maxson,* Milwaukee, Wis.—The authors and management of the Tennessee Copper Co. are to be congratulated for their willingness to have the progressive improvement in their milling practice made available.

The paper might well be considered from the standpoint of the change that has taken place in the adaptation of the rod mill since its introduction at the Detroit Copper Co. about 1914.

During its initial development it was considered as a competitor to the existing Huntington mills, but the field was broadened subsequently to place the rod mill in a commanding position in the field of grinding, where it was desired to produce a relatively granular product.

As often happens, some applications were made that were not economically justifiable; specifically, those where the rod mill was used as a fine grinder beyond its economical range. This is illustrated by the old practice cited in the paper.

The present practice at Copper Hill represents an entirely proper type of installation; namely, the use of the rod mill as a combined fine crusher and coarse grinder.

When used in this manner it may be considered as a multiple set of rolls, but without the attendant continual adjustment and maintenance that may be required on roll installations.

Obviously, its field is somewhat broader than rolls, in that it will produce a finer product economically.

There has been a marked tendency in many large milling operations to continually reduce the size of the feed to the grinding circuit, and the rod mill offers some advantages as compared with the conventional type of crushing machines, primarily because it can be extended to a finer range, and because it eliminates the necessity for closed-circuit screenings and maintenance of crushing equipment at close settings, which is always costly.

The record shows that this can be accomplished on a reasonably hard ore with a decrease in operating cost.

* Allis-Chalmers Manufacturing Company.

It is not, however, to be expected that applications of this type can be extended to all classes of ores, although some very recent installations indicate that the rod mill can be used economically for ores that are considerably harder than those of the Ducktown basin.

This is a variable that must be interpreted by actual testing. However, it is susceptible of quick and reasonably accurate determination.

With reference to the actual operation of the mills, it is significant that there was a marked decrease in steel consumption as the mill speeds were increased, and as the size of the feed was likewise increased. Another point of importance is the gain in capacity, which was secured by two-stage operation as compared with single-stage operation.

This is somewhat obscured by the fact that balls were substituted in the secondary mills, but by using the rod mill as a coarse grinder the ball-mill operation was made quite efficient because it was possible to decrease the size of the balls and thereby increase the efficiency of the grinding mass.

The most significant fact remains, however, that a marked increase in tonnage was secured with practically the same plant equipment, and this was accompanied by a decrease in operating cost, which is important in all milling operations.

O. H. Johnson,* Denver, Colo.—It was interesting to know that Messrs. Myers and Lewis had speeded up their rod mills from the conventional 17 r.p.m., for a 6 by 12-ft. machine, to 19 r.p.m. and finally to 22 r.p.m.

Marcy rod mills at the International Nickel Company of Canada were speeded up more than two years ago, to 23 r.p.m. These machines are 7 ft. in diameter and running about 77 per cent of critical speed. This is much faster than the mills mentioned by Myers and Lewis, which run at only about 70 per cent of critical speed. These percentages of critical speed are based on the diameter of mills inside the linings.

Our first effort toward stepping up the speed of Marcy mills was at Ajo, Ariz., where we went from 17 to 22 r.p.m. The reason we advocated the higher speeds was to get more capacity, and our observations indicated that as the speeds were stepped up, the power and

* Vice President, The Mine and Smelter Supply Company.

tonnage went up proportionately, making practically the same grind. I think Mr. R. A. Pallanch, Mill Superintendent of the U. S. Smelting, Refining and Mining Co., Midvale, Utah, has had the same experience.

In operations of both ball and rod mills, where we find that the power required is not up to the full capacity of the motor, we frequently increase the speeds, to take advantage of the full power available, and in that way the operator has also been able to get more tonnage with the same installation.

Of course, it must be clearly understood that on such big Marcy mills in use at Ajo and International Nickel, where we have made big increases in speed, it has been necessary to change the motors to meet the new power requirement.

Standard Grindability Tests and Calculations

By Fred C. Bond* and Walter L. Maxson,† Members A.I.M.E.

(New York Meeting, February 1943)

Since the last publication of tabulated results of grindability tests by the authors[1] the total number of ball-mill tests made has more than doubled, and rod-mill tests have become increasingly important. Nearly all of the standard closed-circuit ball-mill and rod-mill tests made to date are included in the present tabulation, together with additional information not previously published.

Standard Ball-mill Grindability Tests

The method of conducting these closed-circuit tests has not been altered. A sample of the ore or other material is stage-crushed in rolls set at ¹⁄₁₆-in. opening with a 6-mesh screen until all of it has passed the screen. The combined minus 6-mesh screen undersize is mixed, sampled and screen-analyzed, and its apparent specific gravity is determined by packing and shaking in a standard container, and weighing. The apparent specific gravity ordinarily is about 60 per cent of the true specific gravity. The unit volume present in the mill in all tests is 700 c.c. of the packed minus 6-mesh roll product, and the number of grams occupying 700 c.c. is the unit test weight.

This weight is placed in the mill dry, ground for the number of revolutions estimated to be necessary, discharged, and screened mechanically in three testing sieves of the mesh size at which the test is to be conducted. The oversize, or circulating load, is weighed, sufficient fresh minus 6-mesh feed is added to bring the total weight up to that of 700 c.c., and the charge is returned to the mill for a second grinding period.

All standard ball-mill tests are conducted at 250 per cent circulating load, and the number of revolutions in the mill necessary to obtain this circulating load at any grinding period is estimated from the results of the previous period. The number of net grams of screen undersize produced per revolution of the mill approaches its final equilibrium value after several grinding periods, and this is recorded as the grindability, or relative ease of grinding, of the sample.

A cylindrical ball mill, 12 by 12 in. inside, with a smooth lining and rounded corners, is used with a revolution counter. Under standard grinding conditions it runs at 70 r.p.m., and contains a charge of 285 iron balls weighing 20,125 grams, ranging from 1½ to ¾ in. in diameter.

Tests are conducted at the mesh size to which the ore is to be ground in practice, and the capacity to be expected from a given mill is calculated by comparing the grindability with that of a similar standard ore whose performance characteristics are known.

No satisfactory general formula for converting standard grindabilities into commercial power-consumption values has been

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[1] F. C. Bond and W. L. Maxson: Grindability and Grinding Characteristics of Ores. *Trans.* A.I.M.E. (1938).

found, since considerable judgment may be required in correlating such diverse factors as size of feed, varying circulating loads, stage grinding, presence of natural grain sizes and slimes, and mill speeds.

Caution should be used in making comparisons between ores that are widely dissimilar in nature, as well as between those with widely different grindabilities. If these limitations are observed, the standard grindability test supplies a relatively accurate method for predicting grinding results.

Table 1 gives the condensed results of 374 standard ball-mill grindability tests. Those for each mesh size are arranged in the order of increasing ease of grinding, so that the list for any mesh constitutes an ascending graduated scale of grindabilities.

The grindabilities of ores on which tests have been made at more than one mesh size are marked by an asterisk (*), and those on which standard rod-mill tests have also been made are marked by a dagger (†). In addition to the grindability at the specified mesh, the percentage of minus 200-mesh in the ball-mill product, the apparent and true specific gravities, and the reference test number are included in the tabulation.

If the grindabilities of the ores that have been tested at several different mesh sizes are plotted as abscissas on a logarithmic scale, against the mesh sizes at equal intervals on a linear scale, it is seen that in general the grindabilities of different ores follow parallel straight lines.

However, some ores with an important natural grain size, such as sandstones, have a much steeper slope than the average at mesh sizes larger than their natural grain size. The existence of natural grain sizes has considerable effect upon the plotted grindabilities, so that the grindabilities at different meshes cannot be predicted with dependable accuracy from a test made at only one mesh size.

Standard Rod-mill Grindability Tests

Standard closed-circuit rod-mill grindability tests are made in a mill 12 in. in inside diameter by 22 in. long, with a wave type of lining, and running at 46 r.p.m. The grinding charge consists of eight steel rods 21 in. long, weighing 33,380 grams. Six of the rods are 1¼ in. in diameter and two are 1¾ in. The mill is provided with a revolution counter.

The mill can be tilted through a complete circle by means of a tilting wheel. It is discharged through a trunnion bearing and a grate, which retains the rods when the mill is tilted downward. At intervals of 10 revolutions during a standard test it is tilted 5° upward for one revolution, then 5° downward for one revolution, and returned to the horizontal position for eight revolutions. If this is not done the test results are apt to be erratic, because of material that escapes grinding by lodging between the ends of the rods and the ends of the mill. After the grinding period has been completed, the mill is discharged by tilting it downward at an angle of 45° for 30 revolutions.

The unit volume present in the mill in all tests is 1250 c.c. of dry solids packed by shaking, and the number of grams occupying 1250 c.c. is the unit test weight. Unless the material is exceptionally soft, the feed should not be coarser than ½ in. All standard rod-mill grindability tests are conducted at 100 per cent circulating load.

The weight occupying 1250 c.c. packed is placed in the mill dry, rotated for the number of revolutions estimated to be necessary to produce 50 per cent of screen undersize, discharged, and screened at the mesh size at which the test is to be made. The oversize, or circulating load, is weighed, sufficient fresh feed is added to bring the total weight up to that of 1250 c.c., and the charge is returned to the mill for the second grinding period.

TABLE 1.—*Standard Ball-mill Grindability Tests*

Ore	Description by	Location	Net Grams Undersize per Revolution	−200-mesh in Product, Per Cent	Specific Gravity		No.
					Apparent	True	
Tests at Minus 28-Mesh							
			−28M.				
Gold	Bernheim	S. Rhodesia	1.26*	27.9	1.77		477
Gold	Wright-Hargreaves	Ontario	1.338*	34.2	1.70	2.68	406
Gravel	Pacific Coast Aggregates	California	1.58*	25.2	1.53		324
Copper	Amygdaloid, C. R.	Michigan	1.76*	31.0	1.87	2.93	1000
Gold	Portland	Colorado	1.966*	27.2	1.63	2.69	
Gold	Kerr-Addison	Ontario	2.203*	32.9	1.81		799
Gold	La Luz	Nicaragua	2.48*	30.0	1.79		932
Copper	New Cornelia	Ajo, Ariz.	2.50*	30.4	1.63	2.68	684
Gold	Little Long Lac	Ontario	2.726*	40.2	1.69	2.64	570
Gold	San Luis	Mexico	2.905*	20.5	1.67		730
Copper	Shale—White Pine	Michigan	3.08*	33.3	1.79	2.97	1000
Copper	Quincy	Michigan	3.12*	21.0	1.98		1036
Gold	Rand-Springs Mine	S. Africa	3.22*	23.0	1.78	2.71	504
Gold	Kerr-Addison	Ontario	3.365*	32.7	1.83		799
Copper	Miami Ore	Miami, Ariz.	3.37*	26.9	1.64	2.69	
Quartz	Can. Silica Products	Quebec	3.58	11.7	1.56	2.65	398
Copper	Castle Dome Ore	Miami, Ariz.	3.655*	24.8	1.685		1042
Copper	Utah Copper—Arthur	Utah	3.91*†	26.5	1.715		938
Gold	Homestake	S. Dakota	3.95*	35.5	1.91	3.12	
D.-L.[a] sinter	Eagle Picher	Illinois	4.01	21.1	2.25		341
Silver	Hualgayoc	Peru	4.27	25.2	2.45		346
Copper	Sandstone—Vein Rock	Michigan	4.403*†	20.7	1.83	2.68	1000
Copper	Anaconda	Montana	4.44*	34.3	2.18	3.23	910
Quartz	Pure Crystallized	California	4.55*	17.0	1.68	2.65	
Magnetite	M. A. Hanna, Clifton	New York	4.57†	14.8	2.76		1022
Cassiterite	Vulcan Detinning	New Jersey	5.09	23.1	3.60		558
Gold	East Malartic	Quebec	5.42*	37.8	1.87	2.79	779
Syenite	Can. Nepheline Syenite	Ontario	5.51†	16.7	1.63	2.73	663
Magnetite	Republic Steel, Chateaugay	New York	6.265*	8.8	2.98		868
Copper	Morenci	Arizona	6.34*†	24.8	1.57	2.63	913
Copper	Sandstone—White Pine	Michigan	8.63*	13.8	1.70	2.635	1000
Phosphate	Charleston Min. Co.	Tennessee	8.87†	14.3	1.86		333
Roasted	Sullivan Min. Co.	Idaho	15.33*	38.9	1.93		740
Gold	Red Cross Min. Co.	California	20.9	26.7	1.56		331
Tests at 35-Mesh							
			−35M.				
Cast iron	Du Pont	Delaware	0.062	8.6	3.12		325
Graphite	Alcoa	Tennessee	0.24	25.4	1.01		937
Cast iron	Crucible Steel	Pittsburgh	0.611	2.6	0.94		436
Copper	Conglomerate, C. & H.	Michigan	1.231*	33.5	1.69		771
Gravel	Pac. Coast Aggregates	California	1.28*	28.2	1.53		324
Copper	Amygdaloid, C. R.	Michigan	1.53*†	36.5	1.87	2.93	1000
Copper	Amygdaloid, C. & H.	Michigan	1.82*	30.3	1.75		771
Gold	La Luz	Nicaragua	2.08*	36.0	1.79		932
Gold	Kerr-Addison	Ontario	2.17*	37.3	1.83		799
Copper	Tailings—Quincy	Michigan	2.18*	28.9	1.98		1036
Copper	New Cornelia	Ajo, Ariz.	2.31*	34.7	1.63	2.68	684
Gold	Little Long Lac	Ontario	2.35*	32.6	1.69	2.64	570
Gold	Benguet	P.I.	2.38*	35.6	1.69	2.66	550
Gold	Rand-Springs Mine	So. Africa	2.404*†	27.7	1.78	2.71	504
Gold	San Luis	Mexico	2.495*	25.7	1.67		730
Copper	Shale—White Pine	Michigan	2.66*	38.2	1.79	2.97	1000
Gold	Kerr-Addison	Ontario	2.95*	37.6	1.81		799
Iron	Bowring	Morocco	2.97*	25.0	1.86		288
Copper	Castle Dome Ore	Miami, Ariz.	2.99*	29.5	1.68		1042
Iron	Bowring	Spain	3.03*	20.9	2.12		291
Copper	Anaconda	Montana	3.31*	29.3	2.18	3.23	910
Copper	Utah Copper—Arthur	Utah	3.91*†	30.0	1.71		938
Copper	Sandstone—Vein Rock	Michigan	4.05*†	28.8	1.83	2.68	1000
Copper	Morenci	Arizona	4.50*†	27.5	1.57	2.63	913
Copper	Sandstone—White Pine	Michigan	4.51*	20.4	1.70	2.64	1000
Iron	Republic Steel, Chateaugay	New York	4.70*	12.3	2.98		868
Iron	Alan Wood Steel, Croton	New York	5.35*	16.2	2.18		931
Clinker	Cobrecite	Michigan	7.10*	12.6	0.76		830
Roasted	Sullivan Mines	Idaho	8.04*	40.8	1.93		740

[a] Dwight-Lloyd.
* Standard ball-mill tests made at other mesh sizes.
† Standard rod-mill tests made on this ore.

TABLE 1.—(*Continued*)

Ore	Description by	Location	Net Grams Undersize per Revolution	−200-mesh in Product, Per Cent	Specific Gravity: Apparent	Specific Gravity: True	No.
TESTS AT 48-MESH							
			−48M.				
Petr. coke	Alcoa	Tennessee	0.304	26.9	0.97	1.78	700
Scrap emery	Milwaukee Steel	Wisconsin	0.408*†	40.3	1.93		989
Zinc	National Zinc	Oklahoma	0.572	23.7	0.97		161
Refractory	Laclede Christy	Illinois	0.86*	24.4	1.65		941
Gold	Bernheim	S. Rhodesia	0.87*	42.7	1.77		477
Flint	Tri-State Flint	Missouri	0.90*	29.4	1.54		440
Gold	Wright-Hargreaves	Ontario	1.24*	44.0	1.70	2.68	406
Copper	Amygdaloid, C. R.	Michigan	1.37*†	42.3	1.87	2.93	1000
Gold	Kelowna Expl.	B. Columbia	1.37*	47.5	2.09	3.22	676
Gold	Noranda	Quebec	1.47*	44.9	1.85	2.86	
Gold	Portland	Colorado	1.64*	36.7	1.63	2.69	
Brick mix	Carnegie Steel	Illinois	1.82	24.5	1.54		907
Gold	Little Long Lac	Ontario	1.826*	41.4	1.693	2.64	570
Gold	La Luz	Nicaragua	1.855*	41.0	1.79		932
Copper	Tailings—Quincy	Michigan	1.950*	35.7	1.98		1036
Gold	A. O. Smith Corp., R. M.	Nevada	1.96	34.9	1.61		646
Firebrick	Carnegie Steel	Illinois	1.98	24.5	1.49		907
Gold	Rand-Springs Mine	S. Africa	1.985*	35.9	1.78	2.71	504
Silver	Real Del Monte	Pachuca	2.00*	46.1	1.63		881
Copper	New Cornelia	Ajo, Ariz.	2.10*	40.0	1.63	2.68	684
Gold	San Luis	Mexico	2.123*	27.7	1.67		730
Gold	San Fernando	Mexico	2.28	34.6	1.64		255
Copper	Shale-White Pine	Michigan	2.296*	45.4	1.79	2.97	1000
Gold	San Fernando	Mexico	2.31	35.8	1.56		255
Gold	Saramarca	Peru	2.39	32.4	1.82		547
Garnet	K. T. Felder	Georgia	2.40	36.5	1.78		269
Phosphate	Al-Ke-Me Fertilizer	Brazil	2.41*	32.9	1.39		252
Copper	Castle Dome Ore	Miami, Ariz.	2.52*	36.1	1.68		1042
Copper	Miami Ore	Miami, Ariz.	2.61	38.2	1.64	2.69	
Lead-zinc	Montecatini	Italy	2.69*	31.4	2.32		469
Quartz	Pure Crystallized	California	2.72*	27.1	1.68	2.65	
Copper	Anaconda	Montana	2.75*	34.4	2.18	3.23	910
Gold	Homestake	S. Dakota	2.95*	45.1	1.91	3.12	
Gold	Picacho Min. Co.	Arizona	2.956*	31.4	1.52		880
Copper	Sherritt Gordon	Manitoba	2.97	31.2	1.86		
Copper	Cons. Copper Co.	Nevada	3.00	35.0	1.54		188
Copper	Sandstone—White Pine	Michigan	3.05*	28.5	1.70	2.64	1000
Spodumene	Solvay Process	New York	3.15	23.1	1.73		1073
Copper	Sandstone—Vein Rock	Michigan	3.15*†	28.7	1.83	2.68	1000
Copper	Utah Copper—Arthur	Utah	3.23*†	36.9	1.72		938
Copper	Morenci	Arizona	3.33*†	33.8	1.57	2.63	913
Shale	St. Lawrence Brick	Quebec	3.50*	40.0	1.59		279
Gold	East Malartic	Quebec	3.698*	37.8	1.87	2.80	779
Copper	Silver Bell	Arizona	3.74*	37.1	1.69		1050
Gold	C. M. & R.—Golden Rose	B. Columbia	3.77*	40.5	2.10		864
Gold	S. A. Devel. Co.	Ecuador	4.08*	38.7	2.67	3.81	752
Iron	Alan Wood Steel, Croton	New Jersey	4.34*	21.6	2.18		931
Copper	Tennessee Copper	Tennessee	4.80*	30.2	2.08		263
Copper	Cons. Copper Co.	Nevada	5.32	38.2	1.64		189
Magnesium	Basic Refr. Inc.	Ohio	5.465*	48.4	2.134	2.927	1033
Al sinter	Aluminum Co. of Canada	Quebec	5.70*†	37.4	0.83		1080
Copper	Bingham Canyon, U. S. M.	Utah	5.90*	30.7	2.68	4.65	
Iron	Inland Steel, B. R. F.	Wisconsin	6.04*	30.2	2.11		837
Zinc	Farrey Min. & Mill. Co.	Illinois	6.74		1.74		184
Graphite	C. A. Condon	Alabama	9.26	24.5	1.35		287
Tripoli	Western Minerals	Kansas	12.33†	75.5	1.14		883
TESTS AT 65-MESH							
			−65M.				
Cast iron	Miami Copper	Arizona	0.044	29.3	3.51	7.07	372
Chromium metal	Electro Met. Co.	New York	0.313*†	10.9	3.91		1008
Emery	Am. Emery Wheel Wrks.	Rhode Island	0.459†	51.2	2.47	3.89	410
Copper	Amygdaloid, C. R.	Michigan	1.180*†	50.5	1.87	2.93	1000
Gold	Kelowna Expl.	B. Columbia	1.24*	54.4	2.09	3.22	676
Gold	Little Long Lac	Ontario	1.49*	50.8	1.69	2.64	570
Silver	Real Del Monte	Pachuca	1.53*	45.6	1.63		881
Gold	La Luz	Nicaragua	1.55*	50.6	1.79		932
Copper	Tailings—Quincy	Michigan	1.575*	47.0	1.98		1036

* Standard ball-mill tests made at other mesh sizes.
† Standard rod-mill tests made on this ore.

TABLE I.—(*Continued*)

Ore	Description by	Location	Net Grams Undersize per Revolution	−200-mesh in Product, Per Cent	Specific Gravity		No.
					Apparent	True	
TESTS AT 65-MESH—(*Continued*)							
			−65M.				
Copper.........	Britannia Min. & Sm.	B. Columbia	1.577	39.1	1.71		326
Gold...........	A. O. Smith	Wisconsin	1.583	54.2	1.60		587
Coal...........	Illinois Zinc	Illinois	1.61	47.8	1.69	2.80	376
Copper.........	Sandstone—White Pine	Michigan	1.623*	42.2	1.70	2.64	1000
Sea shells.......	Calizos	Chile	1.637	43.8	1.52		967
Gold...........	Rand-Springs Mine	S. Africa	1.717*	44.7	1.785	2.71	504
Gold...........	Madsen Red Lake	Ontario	1.730*	40.5	1.70	2.65	626
Gold...........	G. E. Smith	Oregon	1.85	37.9	1.60		321
Granite.........	Picacho	Arizona	1.877	39.3	1.53	2.64	517
Gold...........	Spring Hill	Calif.	1.903	45.9	1.70		538
Copper.........	Shale—White Pine	Michigan	1.936*	54.2	1.79	2.97	1000
Iron............	Bowring	Morocco	1.94*	36.8	1.86		288
Copper.........	New Cornelia	Ajo, Ariz.	2.022*	45.0	1.63	2.68	684
Copper.........	Castle Dome	Miami, Ariz.	2.023*	44.9	1.685		1042
Copper.........	Chelan Copper	Washington	2.07	44.5	1.64		328
Copper.........	Cons. Copper Co.	Nevada	2.10	43.7			278
Copper.........	Cyprus Mines	Cyprus	2.110*	43.4	2.69		432
Copper.........	Stadacona Rouyn	Quebec	2.157	53.2	1.89		561
Serpentine......	G. E. Baker Co.	Pennsylvania	2.246	64.2	1.70		1051
Phosphate......	Ipanema Plant	Brazil	2.255*	42.8	2.36		352
Gold...........	Picacho Min. Co.	Arizona	2.26*	46.4			880
Gold...........	Cline Lake	Ontario	2.29*	54.7	1.78		745
Manganese......	C. L. Walfred	Minnesota	2.30*	50.7	1.64		867
Gold...........	Santa Maria del Oro	Mexico	2.31*	33.3	2.08	3.10	574
Gold...........	LaLuz	Nicaragua	2.32*	44.7	1.79		932
Iron............	Moose Mountain	Ontario	2.33*†	53.9	2.44	3.42	756
Copper.........	Sandstone, C. R.	Michigan	2.34*†	37.4	1.83	2.68	1000
Lead-zinc.......	Montecatini	Italy	2.39*	42.6	2.32		469
Gold...........	Parcoy	Peru	2.393*	31.3	2.26		567
Gold...........	Minnesota Mines	Colorado	2.420*	43.0	1.85	2.81	637
Copper.........	Anaconda	Montana	2.435*	41.2	2.18	3.23	910
Copper.........	Morenci	Arizona	2.52*†	44.9	1.57	2.63	913
Gold...........	Homestake	S. Dakota	2.55*	53.7	1.91	3.12	
Gold...........	Ipo Mine	P.I.	2.61*	37.2	1.68		329
Copper.........	Utah Copper—Arthur	Utah	2.65*†	46.6	1.715		938
Gold...........	A. O. Smith	Wisconsin	2.675	55.0	1.60		587
Salt............	M. D. P. d'Alsace	France	2.685*†	22.9	1.32		726
Tungsten........	Nevada-Mass. Co.	Nevada	2.69†	37.0	1.71		942
Copper.........	Silver Bell	Arizona	2.80*	46.0	1.68		1050
Iron............	Alan Wood Steel, Croton	New Jersey	2.86*	31.2	2.18		931
Gold...........	S. A. Devel. Co.	Ecuador	3.02*	50.8	2.67	3.81	752
Limestone.......	H. J. Kaiser Co.	California	3.123	45.5	1.66		877
Gold...........	East Malartic	Quebec	3.14*	58.1	1.87	2.79	779
Copper.........	Tennessee Copper	Tennessee	3.29*	40.9	2.08		263
Magnesite......	N. W. Magnesite Co.	Washington	3.370*	38.6	3.09		1031
Copper.........	Cons. Copper	Nevada	3.51	46.0	1.64		276
Slag............	Monsanto Chem.	Alabama	3.81*	17.3	1.90		1045
Iron............	Alan Wood Steel	New Jersey	4.15†	36.6	2.41	3.40	914
Fe-Si, 25-75.....	Electro-Met. Co.	New York	4.17*	33.9	1.94		1067
Al sinter........	Aluminum Co. of Canada	Quebec	4.23*†	36.0	0.83		1080
Clay............	H. J. Kaiser Co.	California	4.235	32.7	1.36		877
Magnesium.....	Basic Magnesium	Nevada	4.270*	54.0	2.13	2.93	1033
Fe-Mn alloy.....	Champion Rivet Co.	Ohio	4.523	35.0	4.63		442
Fluorspar.......	Aluminum Ore Co.	Illinois	5.700	35.0	1.88	2.98	619
Clinker.........	Cobrecite	Michigan	6.82	17.5	0.76		830
Pyrite..........	St. Joseph—Conc.	New York	21.7	30.7	2.99		847
Bauxite.........	Swann and Co.	Alabama	51.2	23.3	1.08		738
TESTS AT 100-MESH							
			−100M.				
Petr. coke.......	Alcoa	Tennessee	0.19	56.0	1.01		700
Cr Metal.......	Electro-Met. Co.	New York	0.2115*†	21.2	3.91		1008
Gold...........	Chas. Butters	Nicaragua	0.393*	56.7	1.53		338
Flint...........	Tri-State Flint	Missouri	0.618*	52.6	1.54		440
Gold...........	Bernheim	S. Rhodesia	0.79*	60.0	1.77		477
Gold...........	Veraguas Mines	Panama	0.832	71.8	1.75	2.78	498
Copper.........	Conglom.—C. &. H.	Michigan	0.833	63.4	1.69		771
Abrasive........	Monsanto Chem.	Alabama	0.858	30.2	2.11	3.91	656
Fire clay........	Standard Fuel	Michigan	0 964*	46.8	1.25		835
Gold...........	Wright-Hargreaves	Ontario	0 98*	61.0	1.70	2.68	406

* Standard ball-mill tests made at other mesh sizes.
† Standard rod-mill tests made on this ore.

TABLE 1.—(*Continued*)

Ore	Description by	Location	Net Grams Undersize per Revolution	−200-mesh in Product, Per Cent	Specific Gravity		No.
					Apparent	True	
TESTS AT 100-MESH—(*Continued*)							
			−100M.				
Gold	Kelowna Expl.	B. Columbia	1.01*	67.5	2.09	3.22	676
Gold	Western Mach. Co.	Calif.	1.040	59.4	1.62		903
Gold	Noranda	Quebec	1.079	69.9	1.85	2.86	
Gold	Little Long Lac	Ontario	1.150*	67.8	1.69	2.64	570
Gold	Wright-Hargreaves	Ontario	1.15*	67.7	1.70	2.68	406
Copper	Amygdaloid—C. & H. Min.	Michigan	1.185*	66.1	1.745		771
Silver	Real Del Monte	Pachuca	1.230*	56.5	1.63		881
Copper	Sandstone—White Pine	Michigan	1.250*	55.6	1.70	2.64	1000
Gold	Portland	Colorado	1.256	56.6	1.63	2.69	
Gold	Amer. Cyanamid	New Jersey	1.288	62.7	1.68	2.61	524
Sp. iron	Ford Motor	Michigan	1.295*	46.8	3.48		1035
Gold	LaLuz	Nicaragua	1.30*	63.4	1.79		932
Gold	Rand-Springs Mine	S. Africa	1.323	62.3	1.78	2.71	504
Gold	W. A. Liddell	Texas	1.335	46.3	1.75	2.59	632
Fire clay	Standard Fuel	Michigan	1.355*	52.4	1.23		835
Phosphate	Al-Ke-Me	Brazil	1.36	61.8	1.39		252
Gold	San Luis	Mexico	1.41*	49.7	1.67		730
Copper	Tailings—Quincy	Michigan	1.413*	59.3	1.98		1036
Gold	E. T. Merritt	Ontario	1.46	55.8	1.64		301
Phosphate	Ipanema Plant	Brazil	1.47*	59.6	2.36		352
Copper	Kanshanshi	Congo	1.49	62.9	1.32		316
Silver	Cia. Min. Carlota	Chile	1.51*	72.0	1.88	3.00	660
Gold	Madsen Red Lake	Ontario	1.515*	53.4	1.70	2.65	626
Quartz	Pure Crystallized	Calif.	1.523*	53.0	1.68	2.65	
Gold	Getchell Mine	Nevada	1.535	59.3	1.45		714
Silver	Tonopah Min.	Nevada	1.54	62.35	1.77		802
Copper	Shale—White Pine	Michigan	1.56*	70.6	1.79	2.97	1000
Copper	New Cornelia	Ajo, Ariz.	1.57*	73.1	1.63	2.68	684
Gold	Santa Maria Del Oro	Mexico	1.575*	45.7	2.08	3.10	574
Copper	Sandstone—Vein Rock	Michigan	1.577*†	56.7	1.83	2.68	1000
Gold	Dahlonega Gold	Georgia	1.58	51.9	1.67		250
Gold	Sherritt Gordon	Manitoba	1.64	57.0	1.85		206
Gold	Am. Metals	New Jersey	1.676*	50.0	1.74		808
Manganese	Gen. Manganese Co.	S. Dakota	1.71		1.62		734
Gold	Ziebright Mine	Calif.	1.752	56.4	1.79		505
Copper	Castle Dome	Miami, Ariz.	1.783*	56.1	1.69		1042
Copper	Cyprus Mines	Cyprus	1.786*	60.3	2.69		432
Tin	Tainton Products	Bolivia	1.80	52.1	2.42		905
Gold	F. Viles	Montana	1.81	63.9	1.72		564
Copper	Miami Ore	Miami, Ariz.	1.816*	52.2	1.64	2.69	
Manganese	C. C. Walfred	Minn.	1.83	60.0	1.69		867
Copper	Anaconda Copper	Montana	1.85*	54.7	2.18	3.23	910
Gold	Rochester-Plymouth	Nevada	1.860*	63.1	1.83	2.75	518
Copper	Morenci	Arizona	1.88*†	54.5	1.57	2.63	913
Gold	Cline Lake	Ontario	1.92*	67.3	1.78		745
Gold	Homestake	S. Dakota	1.964*	66.5	1.91	3.12	
Copper	Silver Bell	Arizona	1.98*	59.6	1.68		1050
Gold	Atlantic Gulf & Pacific	P.I.	2.01*	52.9	1.68		329
Copper	Utah Copper—Arthur	Utah	2.15*†	57.8	1.72		938
Slag	Monsanto Chem.	Alabama	2.16*	31.6	1.90		1045
Tin	Pitts. Plate Glass	Mexico	2.226*	55.5	2.92		1018
Phosphate	Int. Agric. Corp.	Florida	2.260	47.3	1.50		394
Fe-Cr alloy	Chromium Min. & Smelt.	Ontario	2.27	47.8	3.97		743
Tin	Pitts. Plate Glass	Mexico	2.335*	56.7	3.04		1018
Nickel	Falconbridge	Ontario	2.405	59.3	2.29	3.65	371
Magnesite	NW. Magnesite Co.	Washington	2.410*	50.3	3.09		1031
Gold	S. Amer. Dev. Co.	Ecuador	2.44*	63.1	2.67	3.81	752
Gold	E. Malartic	Quebec	2.49*	68.0	1.87	2.80	779
Iron	Marquette Carbonate	Wisconsin	2.50	79.2	2.17		342
Gold	C. M. & S.—Golden Rose	B. Columbia	2.52*	58.2	2.10		864
Copper	Cons. Copper	Nevada	2.59*	60.2	1.64		276
Iron	Bowring, N. Y.	Morocco	2.66*	49.4	1.80		315
Iron	Iron River Falls	Wisconsin	2.72*	58.7	2.39		1023
Fe-Si, 25–75	Electro-Met. Co.	New York	2.950*	47.2	1.94		1067
Copper	Bingham Canyon	Utah	2.99	42.6	2.68	4.65	
Gold	Butte-Highlands	Montana	3.13*	58.3	1.67		861
Salt	Du Pont	New York	3.48	44.8	1.39		939
Magnesium	Basic Refr. Inc.	Ohio	4.125*	61.9	2.13	2.93	1033
Slag	Victor Chem. Wks.	Tenn.	4.15	48.0	4.05		284

* Standard ball-mill tests made at other mesh sizes.
† Standard rod-mill tests made on this ore.

TABLE 1.—(*Continued*)

Ore	Description by	Location	Net Grams Undersize per Revolution	−200-mesh in Product, Per Cent	Specific Gravity		No.
					Apparent	True	
Tests at Minus 150-mesh							
			−150M.				
Graphite........	Long Valley—Cons.	New York	0.31*	49.8	0.96		860
Copper.........	Sandstone—White Pine	Michigan	0.85*	84.1	1.70	2.64	1000
Gold...........	Kelowna Expl.	B. Columbia	0.86*	77.9	2.09	3.22	676
Quartz.........	Fused Quartz sand	Illinois	1.02	63.5	1.48		854
Gold...........	Little Long Lac	Ontario	1.080*	82.6			570
Gold...........	Rand Springs Mine	S. Africa	1.117*	76.7	1.78		504
Gold...........	H. C. Winans	Brazil	1.155	76.8	1.63		474
Gold...........	Powell-Rouyn	Quebec	1.233*	79.5	1.78		949
Copper.........	Sandstone—White Pine	Michigan	1.249*	73.1	1.78		1000
Gold...........	Bong Mieu	Indo-China	1.31	75.7	2.00	3.07	339
Gold...........	Kerr-Addison	Ontario	1.32*	81.6	1.81		799
Copper.........	Anaconda	Montana	1.36*	75.2	2.18	3.23	910
Gold...........	Minnesota Mines	Colorado	1.368*	77.6	1.85	2.81	637
Copper.........	Morenci	Arizona	1.395*†	77.2	1.57	2.63	913
Copper.........	Castle Dome	Miami, Ariz.	1.396*	78.7	1.69		1042
Copper.........	Mines De Bor	Yugoslavia	1.41*	84.5	1.91		249
Copper.........	Shale—White Pine	Michigan	1.430*	83.7	1.79	2.97	1000
Gold...........	Can.-Malartic Min. Co.	Quebec	1.445	78.5	1.84		586
Gold...........	Parcoy	Peru	1.495*	72.8	2.26		567
Gold...........	Kerr-Addison	Ontario	1.567*	79.3	1.83		799
Nickel.........	B. C. Nickel Mines	B. Columbia	1.607	71.4	2.47		716
Langbeinite.....	Union Potash & Chem.	New Mexico	1.609	77.0	1.75		1006
Copper.........	Utah Copper—Arthur	Utah	1.62*†	77.1	1.72		938
Tin............	Pitts. Plate Glass	Mexico	1.675*	74.6	2.92		1018
Iron...........	Moose Mountain	Ontario	1.689*†	81.5	2.44	3.42	756
Gold...........	Buffalo-Ankerite	Ontario	1.705	82.9	1.94	3.22	614
Gold...........	M. A. Smith	Cuba	1.725	78.2	1.86		710
Tin............	Pitts. Plate Glass	Mexico	1.783*	76.2	3.03		1018
Zinc-lead.......	Callahan Zinc-Lead	Idaho	1.81	60.4	2.23		691
Limestone.......	Lawrence Cement Co.	Pennsylvania	1.870*	76.2	1.90		972
Iron...........	Iron River Falls	Wisconsin	1.916*	77.3	2.39		1023
Gold...........	Preston East Dome	Ontario	2.01*	86.8	1.78		694
Gold...........	C. M. & S.—Golden Rose	B. Columbia	2.03*	74.1	2.10		864
Iron...........	Inland Steel Co.	Wisconsin	2.09*	62.8	2.10		837
Gold...........	E. Malartic	Quebec	2.134*	83.5	1.87	2.80	779
Gold...........	Atienda	Italy	2.84	84.4	1.42	2.56	629
Magnesium.....	Basic Magnesium	Nevada	2.855*	79.5	2.13	2.93	1033
Nickel..........	Nicaro Nickel Co.	Cuba	3.48	73.8	1.13		869
Nickel..........	Nicaro Nickel Co.	Cuba	5.55	86.0	1.29		1075
Tests at Minus 200-mesh							
			−200M.				
Graphite........	Long Valley—Conc.	New York	0.23*		0.96		860
Silicon carbide...	Electro-Ref.	New York	0.259		1.89		1047
Flint...........	Tri-State Flint	Missouri	0.491*				440
Gold...........	Bernheim	S. Rhodesia	0.56*		1.77		477
Shale..........	Korite Corp.	Wisconsin	0.59		1.59		677
Silicon carbide...	Electro-Ref.	New York	0.614		1.25		1047
Titanium.......	Titanium Corp.	Arkansas	0.620		2.36		559
Copper.........	Amygdaloid C. & H. Min.	Michigan	0.627*		1.69		771
Sp. Iron........	Ford Motor Co.	Michigan	0.664*		3.48		1035
Min. Wool......	Mineralite Corp.	Penna.	0.678		0.19		399
Gold...........	San Luis	Mexico	0.688*		1.67		730
Iron...........	Mabellite Corp.	New York	0.719		1.74		419
Silver..........	Real Del Monte	Pachuca	0.720*		1.63		881
Copper.........	Sandstone—White Pine	Michigan	0.733*		1.20	2.64	1000
Gold...........	Kelowna Expl.	B. Columbia	0.758*		2.09	3.27	676
Copper.........	Tailings—C&H Min.	Michigan	0.759*		1.75		771
Gold...........	Wright-Hargreaves	Ontario	0.771*		1.70	2.68	406
Gold...........	Cia. Minera Ciclon	Chile	0.788		1.84		719
Clay...........	Sun Oil Co.	Pennsylvania	0.790		0.66	1.09	641
Gold...........	Carrizalillo	Chile	0.816*		1.74		808
Copper.........	Noranda	Quebec	0.83*		1.85	2.86	504
Gold...........	Rand-Springs Mine	S. Africa	0.859*		1.78	2.71	

* Standard ball-mill tests made at other mesh sizes.
† Standard rod-mill tests made on this ore.

TABLE 1.—(*Continued*)

Ore	Description by	Location	Net Grams Undersize per Revolution	−200-mesh in Product, Per Cent	Specific Gravity		No.
					Apparent	True	

TESTS AT MINUS 200-MESH—(*Continued*)

Ore	Description by	Location	Net Grams Undersize per Revolution	−200-mesh in Product, Per Cent	Specific Gravity Apparent	Specific Gravity True	No.
			−200M.				
Shale	Korite Corp.	Wisconsin	0.86		1.36		677
Quartz	Pure, crystallized	California	0.878		1.68	2.65	
Iron	Du Pont	Pennsylvania	0.89		2.73		251
Gold	Little Long Lac	Ontario	0.903*	1	1.69	2.64	570
Sand	Krebs Pigment	Maryland	0.92		2.79		392
Copper	New Cornelia Ore	Ajo, Ariz.	0.942		1.63	2.68	684
Gold	Galigher Co.	Utah	0.943		1.65		889
Gold	Powell-Rouyn	Quebec	0.960*		1.78		949
Gold	Santa Maria del Oro	Mexico	0.981		2.08	3.10	574
Copper	Anaconda	Montana	0.990*		2.18	3.23	910
Gold	M. A. Smith	Cuba	1.025		1.85		710
Gold	Portland	Colorado	1.035		1.63	2.69	
Copper	Castle Dome	Miami, Ariz.	1.036*		1.69		1042
Silver	Cia. Min. Carlota	Chile	1.042		1.88	3.00	660
Gold	Berens River Mines	Manitoba	1.045		1.95		705
Fe-Si, 25–75	Electro-Met. Corp.	New York	1.075*		1.94		1067
Ti	Vanadium Corp. Am.	Pennsylvania	1.08		2.44		281
Copper	Morenci	Arizona	1.085*†		1.57	2.63	913
Gold	Upper Can. Gold Mines	Ontario	1.097		1.88		821
Manganese	Gen. Manganese Corp.	S. Dakota	1.136		1.62		734
Copper	Miami Ore	Miami, Ariz.	1.139		1.64	2.69	
Gold	Rochester-Plymouth	Nevada	1.141*		1.83		518
Copper	Silver Bell	Arizona	1.152*		1.68		1050
Gold	Kerr-Addison	Ontario	1.153*		1.83		799
Gold	Baguio Gold Min.	P.I.	1.160		1.62	2.63	402
Gold	Kerr-Addison	Ontario	1.175*		1.81		799
Quartz	White Quartz—Sulphides	California	1.20		1.70		725
Gold	Cline Lake	Ontario	1.225*		1.78		745
Iron	Moose Mountain	Ontario	1.227*		2.44	3.42	756
Copper	Utah Copper—Arthur	Utah	1.23*†		1.72		938
Limestone	H. J. Kaiser Co.	California	1.257*		1.66		877
Gold	Homestake	S. Dakota	1.26*		1.91	3.12	
Copper	Cyprus Mines Corp.	Cyprus	1.284*		2.69		432
Copper	Shale—White Pine	Michigan	1.304*		1.79	2.97	1000
Zinc	New Jersey Zinc—Conc.	Penna.	1.315		2.51		849
Gold	So. Amer. Dev. Co.	Ecuador	1.323*		2.67	3.81	752
Lead-zinc	Axerio-Monteponi	Italy	1.396		2.88	4.43	562
Gold	C. M. & S. Co.—Golden Rose	Ontario	1.42*		2.10		864
Zinc	New Jersey Zinc Co.	Penna.	1.482		2.55		503
Clay	H. J. Kaiser Co.	California	1.585*		1.36		877
Gold	E. Malartic	Quebec	1.601*		1.87	2.80	779
Limestone	Lawrence Cement	Pennsylvania	1.667*		1.90		972
Magnesium	Basic Refr. Inc.	Ohio	1.682*		2.13	2.93	1033
Gold	Preston East Dome	Ontario	1.69*		1.78		694
Gold	Butte Highlands	Montana	1.81*		1.66		861
Copper	Bingham Canyon	Utah	1.854		2.62	4.65	
Copper	Mines De Bor	Yugoslavia	1.94		2.1		249
Manganese	L. G. Aguilar & Co.	Cuba	1.944		1.87		1028
Lead slag	Arcade Sm. & Ref.	Mass.	5.16		4.53		360

* Standard ball-mill tests made at other mesh sizes.
† Standard rod-mill tests made on this ore.

The number of revolutions necessary to obtain 100 per cent circulating load, or 50 per cent of screen undersize in the product, is calculated from the results of the previous period. The number of net grams of screen undersize produced per revolution of the mill at 100 per cent circulating load approaches its final equilibrium value after several grinding periods, and this is recorded as the rod-mill grindability, or relative ease of grinding, of the sample.

Tests are conducted at the mesh size to which the ore is to be ground in practice, and the capacity to be expected from a given mill is calculated by comparing the grindability with that of a similar standard ore. As with ball-mill

TABLE 2.—*Standard Rod-mill Grindability Tests*

Ore	Description by	Location	Net Grams Undersize per Revolution	−200-mesh in Product, Per Cent	Specific Gravity Apparent	Specific Gravity True	No.
TESTS AT MINUS 3-MESH							
			−3 M.				
Clinker.........	Volunteer Cement Co.	Tennessee	29.34*	2.82	1.79		828
Granite.........	W. A. Burton	Texas	31.6	2.76	1.60	2.65	875
TESTS AT MINUS 4-MESH							
			−4 M.				
Gravel..........	Warner Co.—Van Sciver	Pennsylvania	22.2	5.71	1.69		815
Iron............	Charleston Iron Min.	Minnesota	25.2	9.2	2.34		754
Clinker.........	Volunteer Cement Co.	Tennessee	26.15*	2.67	1.79		828
Bauxite.........	Republic Min. & Mfg.	Arkansas	37.0	17.50	1.51		1053
TESTS AT MINUS 6-MESH							
			−6 M.				
Gravel..........	Material Service Corp.	Illinois	22.2	8.55	1.60		765
Iron............	Warren Pipe & Foundry	New Jersey	49.5	5.72	2.08		1065
Calcite..........	New England Lime	Mass.	133.6	4.02	1.77		1066
Dolomite.......	New England Lime	Connecticut	319.0	2.39	1.85		1066
TESTS AT MINUS 8-MESH							
			−8 M.				
Silicon-carbide...	Exolon Co.	New York	37.10*	1.92	1.60	3.17	1052
Phosphate......	Federal Chem. Co.	Tennessee	41.50*	24.36	1.80		814
Coal Slag.......	H. B. Reed, Inc.	Indiana	208.5	0.48	1.63		1026
TESTS AT MINUS 10-MESH							
			−10 M.				
Limestone.......	Crushed Rock Prod. Co.	New York	1.63	9.25	1.67		906
Brick...........	Cohart Refr. Inc.	Kentucky	2.65*	16.73	2.22		809
Limestone.......	Pitts. Limestone Corp.	Pennsylvania	9.85	13.0	1.50		723
Gravel..........	Dravo Corp.	Pennsylvania	11.64	12.98	1.64		750
Limestone.......	Franklin Limestone Co.	Tennessee	12.65	15.20	1.65		953
Clinker.........	Volunteer Cement Co.	Tennessee	13.03*	7.72	1.79		828
Glass...........	Corning Glass Wks.	New York	14.15	5.17	1.01		945
Sod. silicate.....	Du Pont	Indiana	16.80	7.61	0.94		954
Chrome.........	Tekirova Madenleri Co.	Turkey	37.40	11.74	2.05		843
TESTS AT MINUS 14-MESH							
			−14 M.				
Radium.........	Eldorado Gold Mines	Ontario	7.76	17.66	1.95	2.77	1063
Tile............	Arketex Ceramic Corp.	Indiana	8.15	18.02	1.60		766
Fluorspar.......	Kinetic Chem. Inc.	New Mexico	11.15*	14.6	1.97		763
Calcines.........	Basic Dolomite Co.		13.50		1.62		583
Feldspar........	Golding Keene Co.	N. Hampshire	14.90*	13.85	1.78		784
Nickel matte....	Int. Nickel Co.	W. Virginia	20.96	13.98	3.81		1009
Langbeinite.....	Union Potash & Chem.	New Mexico	40.90*	8.77	1.59		842
Slag............	Celotex Corp.	Ohio	163.5		0.36		919
TESTS AT MINUS 20-MESH							
			−20 M.				
Graphite........	U.S. Graphite Co.	Michigan	2.22		1.24		728
Brick...........	Cohart Ref. Co.	Kentucky	2.57*	20.61	2.22		809
Iron............	Mozan	Japan	4.20†	20.19	2.41	3.42	914
Alumina........	Exolon Co.	New York	5.19	10.84	2.65		1052
Silicon carbide...	Exolon Co.	New York	7.50	6.73	1.60		1052
Slag............	Ohio Ferro-Alloys	Ohio	8.40	21.82	1.99		933

* Rod-mill tests made at other mesh sizes.
† Standard ball-mill tests made on this ore.

TABLE 2.—(*Continued*)

Ore	Description by	Location	Net Grams Undersize per Revolution	−200-mesh in Product, Per Cent	Specific Gravity Apparent	Specific Gravity True	No.
	TESTS AT MINUS 20-MESH—(*Continued*)						
			−20 M.				
Quartzite	Smith & Koelliker	Ohio	9.26	15.43	1.84		774
Copper	Utah Copper—Arthur	Utah	9.90*†	25.07	1.72		938
Titanium	Nat. Lead Co.	Missouri	11.50*	13.0	2.64		1017
Rutile	American Rutile Corp.	Virginia	11.95	31.75	1.81		971
Feldspar	Golding Keene Co.	N. Hampshire	13.34*	16.30	1.78		784
Copper	Morenci Ore	Arizona	14.98*†	23.75	1.57	2.63	913
Feldspar	Cons. Feldspar Corp.	Tennessee	16.36	14.45	1.52		934
Iron	Rep. Steel—Harmony	New York	18.60†	7.16	2.19		822
Bauxite	Porocel Corp.	Arkansas	21.27	19.60	1.25		884
Iron	Rep. Steel—Old Bed	Mineville, N. Y.	25.95†	7.05	2.70		822
Iron	Rep. Steel—New Bed	Mineville, N. Y.	28.22†	7.05	2.71		822
Barite	Barium Min. Corp.	W. Virginia	59.5	15.7	3.14		1007
	TESTS AT MINUS 28-MESH						
			−28 M.				
Cr Metal	Electro-Met. Co.	New York	1.90†	3.00	3.91	7.01	1008
Sod. silicate	Diamond Alkali Co.	Ohio	4.72	15.56	1.62		739
Glass	Pitts. Plate Glass Co.	Pennsylvania	4.97	10.47	1.49	2.60	746
Manganese	E. W. Creevy	Costa Rica	7.45	24.00	2.02		968
Gypsum	Diamond Crystal Salt		7.77	41.60	1.90		696
Copper	Utah Copper—Arthur	Utah	7.93*†	31.21	1.72		938
Titanium	Nat. Lead Co.	Missouri	9.165*	17.12	2.64		1017
Iron	M. A. Hanna Co.—Clifton	New York	9.30†	15.00	2.76		1022
Copper	Morenci Ore	Arizona	11.98*†	28.86	1.57	2.63	913
Ferrosilicon	Pitts. Met. Co.	New York	20.92*	10.36	2.77		839
	TESTS AT MINUS 35-MESH						
			−35M.				
Pumice	Barnsdall Tripoli Co.	Missouri	4.52	58.4	0.64		749
Zinc	New Jersey Zinc Co.	N. Jersey	5.73	26.0	2.39		930
Phosphate	Federal Chem. Co.	Tenn.	5.87	37.84	1.38		533
Copper	Morenci Ore	Arizona	6.45*†	39.64	1.57	2.63	913
Copper	Utah Copper—Arthur	Utah	6.62*†	35.53	1.72		938
Cold	Mineral Min. Corp.	S. Carolina	7.09†	54.61	1.43		782
Iron	Rep. Steel—Old Bed	New York	9.15*†	18.79	2.70		822
Iron	Rep. Steel—Harmony	New York	9.43*	17.58	2.20		822
Sylvanite	Union Potash & Chem. Co.	New Mexico	11.22*	20.21	1.30		842
Ferrosilicon	Pitts. Met. Co.	New York	13.68*	24.26	2.68		839
Barite	United Pigment—Meggen	N. Jersey	21.01	37.09	2.98		848
Pitch	Crosset Chem. Co.	Arkansas	35.2	26.24	1.27		813
Barite	United Pigment—Tenn.	N. Jersey	73.43	39.94	3.09		848
	TESTS AT MINUS 48-MESH						
			−48M.				
Copper	Amygdaloid	Michigan	1.995†	48.94	1.87	2.93	1000
Copper	Sandstone Vein	Michigan	3.733†	39.40	1.83	2.68	1000
Gold	Seal Harbor Gold Mines	Nova Scotia	4.66	40.4	1.78	2.76	602
Kyanite	Phosphate Recovery Corp.	Virginia	4.77	20.04	1.74	2.82	760
Copper	Morenci Ore	Arizona	4.89*†	44.87	1.57	2.63	913
Fluorspar	Kinetic Chem. Inc.	New Mex.	5.40*	30.5	1.96		763
Sylvanite	Union Potash & Chem.	New Mex.	5.43	28.21	1.23		842
Copper	Utah Copper—Arthur	Utah	5.43*†	43.42	1.72		938
Langbeinite	Union Potash & Chem.	New Mex.	5.73*	31.27	1.59		842
Tripoli	Western Minerals, Inc.	Kansas	6.89†	77.35	1.14		883
Sinter	Aluminum Co. of Can.	Quebec	8.75*†	36.82	0.84		1080
Cryolite	Penna. Salt Mfg. Co.	Pennsylvania	13.93	37.62	1.92		958
	TESTS AT MINUS 65-MESH						
			−65M.				
Fe-Si, 50–50	Electro-Met. Co.	New York	2.92	39.18	3.08		947
Sinter	Aluminum Co. of Can.	Quebec	6.25*†	41.45	0.84		1080

* Rod-mill tests made at other mesh sizes.
† Standard ball-mill tests made on this ore.

grinding, judgment must be exercised in making this comparison.

Table 2 gives the condensed results of 85 standard rod-mill grindability tests, made at mesh sizes ranging from 3 to 65 mesh. The tests made at each mesh size are arranged in the order of increasing ease of grinding. The grindabilities of ores at which rod-mill tests have been made at more than one mesh size are marked by an asterisk (*), and those on which standard ball-mill tests have also been made are marked by a dagger (†). In addition to the grindability at the specified mesh, the percentage of minus 200-mesh in the rod-mill product, the apparent and true specific gravities, and the reference test number are included in the tabulation.

When the rod-mill grindabilities of samples tested at more than one mesh size are plotted on a logarithmic scale against the mesh sizes at equal intervals, the results usually are straight lines. However, there is more variation in the slopes for different materials than in the ball-mill tests, since the mesh sizes are larger and more of the total comminution accomplished is done above the natural grain sizes of the material.

Wear and Size Distribution of Grinding Balls

By Fred C. Bond*

(New York Meeting, February 1940)

The process of comminution by grinding is properly classified as an art, rather than as a science. Like most other operations concerned in ore dressing, or in the treatment of nonmetallic minerals, the mathematical relationships of rock breaking are still obscure, and much disputed. If some basic formula exists, comparable for instance to $E = RI$ in electricity, it has not yet been placed in common use. The absence of such a comprehensive mathematical background undoubtedly is the cause of many inefficient designs and installations, as well as of much poorly oriented experimental work.

On the other hand, the mathematical immaturity of the process constitutes an opportunity for a real contribution on the part of those who enjoy mental pioneering. Some of the statements made in any such exploratory work may be little more than opinions, and it is to be expected that some will later be proved untrue; but such an evolutionary development is necessary if we are ever to arrive at the fundamental laws, the discovery of which can change the empirical art into a mathematically exact science.

The most significant principle of comminution that we possess is probably Rittinger's law, which states that the work done in crushing or grinding is proportional to the area of the new surface produced. There is an alternative theory, known as Kick's law, which formerly was preferred to Rittinger's but which now is rather discredited. According to Kick, the work done in comminution is proportional to the volume, or weight, of the pieces broken, or to the cube of their diameters, while according to Rittinger it is proportional to the square of the diameters.

Careful determinations of size distribution below the limit of sieve sizes[1] indicate strongly that under ordinary grinding conditions the amount of new surface area produced is directly proportional to the net power input. Changes in the size ratio between the balls and the particles cause important differences in the proportion of the product that passes a given mesh size, but do not affect the total amount of new surface area; they merely change the distribution of the surface among mesh sizes.

The principle of constant surface area is the logical result of Rittinger's law, and it may well have an important bearing on the comminution theories of the future.

Formula for Ball Wear

Grinding tests have been made in the Allis-Chalmers laboratory for more than 9 years, using a standard charge of mixed balls of different sizes; and a continuous record has been kept of the rates of wear of each size. These results have been combined and plotted in various ways in order to discover the relationship that exists

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[1] F. C. Bond and W. L. Maxson: Grindability and Grinding Characteristics of Ores. *Trans.* A.I.M.E. (1939) 134, 323.

between the ball size and its rate of wear in a mixed ball charge.

A relationship was discovered eventually that is so consistent for the different ball sizes measured, as well as for observations made over different periods of time and when grinding widely different materials, that there appears to be little doubt of its fundamental nature.

The various ball sizes are given size numbers n, which are calculated from their diameters according to the following formula:

$$n = 6.645(\log d - \bar{9}.9286) + 1 \qquad [1]$$

where d is the ball diameter in centimeters. If d is in microns, the second constant becomes $\bar{5}.9286$.

n is the ordinal size number,[1] or standard screen-size number used for ground products. Particles passing 150 mesh and retained on 200 mesh are No. 41, those passing 100 mesh and retained on 150 mesh are No. 42, and so on. On this scale a ball of 5-in. dia. is No. 61.95. The scale ranges from the approximate atomic dimension as No. 1, to the earth as No. 115.8, and supplies a convenient method for measuring any size reduction.

Our results show that the following relationship exists between the ball size and its rate of wear in a charge of mixed sizes:

$$\text{Log grams weight loss of any ball} = 0.345n + K_1 \qquad [2]$$

where K_1 is a constant for a definite amount of grinding, and n is the ball size. The value of K_1 can be calculated when the weight loss of any one ball of known size in the mill charge is known, and the corresponding weight losses of balls of any other size can be found from the equation.

The value of the coefficient 0.345 was found to be consistent throughout the wide range of grinding conditions encountered in testing, and has considerable significance.

The logarithms of the surface areas of balls of different sizes vary as the log of 2 multiplied by the size number, or as $0.301n$. It is evident from the value of the coefficient in eq. 2 that the rate of wear of the balls is not directly proportional to their surface areas, but that a film of any specified thickness is worn from a large ball slightly faster than from a small ball in the same charge. This would be expected when the greater impact and contact pressure of the larger balls is considered.

Davis assumed[2] that the rate of wear is proportional to the weight, or volume, of the ball. On this basis, the logarithm of the grams of weight loss should vary as the log of 3 multiplied by the size number, or as $0.477n$, which is very different from our measured results. Most published calculations of the proper initial ball charge for a mill are based on Davis' data.

Equilibrium Ball Charge

A mathematical study of the rate of ball wear was undertaken in an attempt to resolve some of the existing uncertainties. The formulas given below were derived by the same methods of reasoning as those published previously regarding the distribution of ground products,[1] and reference to the article cited will make the derivations clear.

It is considered that the grinding mill is operating continuously in an equilibrium condition, and that balls of only one size are fed to replace the ball weight lost by wear. These are usual conditions of ball-mill operation.

The size number of the sieve that would just pass the balls fed to the mill is designated as N_F, and that of the sieve that would just pass the balls discharged from the mill is N_D.

For the purposes of the calculation, it is assumed that all balls discharge from the mill at size 55.5, or when slightly more than ½ in. in diameter, so that $n_D = 55.0$.

In the derivation, N_F is taken as the Y axis and the slope of the distribution line

[2] E. W. Davis: Fine Crushing in Ball Mills. *Trans.* A.I.M.E. (1920) 61, 250–296.

m is taken as the constant 0.345 multiplied by 10. $n_F = N_F - 0.5$, and $n_D = N_D - 0.5$. This difference of ½ unit is caused by the fact that the average diameter of the balls designated as n, which are retained on a sieve of size n and pass the next larger sieve $n + 1$, is midway between the two sieve sizes, while N refers to balls just passing the larger sieve.

P_n is the per cent weight retained on any size n and passing $n + 1$.

P_F is the theoretical per cent weight retained on sieve size N_F, assuming that the distribution line is extended to that size. The largest ball just passes size N_F and its actual value is zero.

The total area beneath the distribution line is 100 per cent; or

$$100 = \int_{x=\frac{N_F - n_D}{10}}^{x=-0.05} P_F(\text{antilog } m)^x \, dx \qquad [3]$$

from which

$$P_F = \frac{79.45}{0.6721 - \dfrac{1}{\text{antilog }[0.345(N_F - 55)]}} \qquad [4]$$

After P_F for the size of ball regularly fed to the mill has been calculated from eq. 4, the size distribution of the ball charge can be obtained from the following equation:

$$\text{Log } P_n = 0.345(n - N_F) + \log P_F \qquad [5]$$

where a difference of one size unit or more exists between N_F and n.

The value of P_n is computed from eq. 5 for each ordinal size number from 55 to the second size below N_F. The percentage retained on the largest size is most conveniently calculated by difference, so that the total is equal to 100 per cent.

The calculated cumulative per cent weight retained on each ordinal size is plotted against the size number, and a smooth curve is drawn through the plotted points. The value of N_F for each standard ball size included in the charge is laid off on the graph, and the cumulative per cent weight retained is found for each size.

This represents the composition of the actual equilibrium ball charge in the mill, after it has been in operation for a reasonable length of time, and new balls of only one size have been added regularly to replace the ball weight lost by wear.

Initial Ball Charge

The initial ball charge used in starting a new mill should have as nearly as possible the same size distribution as that of the equilibrium ball charge, so that the initial grinding results will be similar to those obtained after the ball charge has reached equilibrium. The initial charge must be made up of balls of a limited number of definite sizes, so that at best it can only approximate the distribution of the equilibrium charge, which contains an infinite gradation of ball sizes varying from the maximum size fed down to the reject size.

In order to calculate the proper initial ball charge, we designate the largest ball size used in making up the charge, which is also the size of the balls to be fed continuously to the mill in operation, as b_1, the size next to the largest as b_2, and so on.

The per cent weight of balls of each size in the initial ball charge, which most nearly corresponds to the equilibrium charge, is computed as follows:

Percentage of size b_1 = ½ (per cent cum. on b_2)
Percentage of size b_2 = ½ (per cent cum. on b_3)
Percentage of size b_3 = ½ (per cent cum. on b_4 minus per cent cum. on b_2)
Percentage of size b_4 = ½ (per cent cum. on b_5 minus per cent cum. on b_3)

and so on for all of the ball sizes added. A sufficient amount of the smallest ball size used is added to make the total 100 per cent. The per cent cumulative on each size is found from the plotted values of N_F for that size.

There is always a larger amount of size b_2 present than of size b_1, since the actual

equilibrium charge contains no balls larger than size b_1.

Calculated initial ball charges for mills fed continuously with each of the common ball sizes from 5 in. down are given in Table 1, as are also the equilibrium ball charges and the constants used in making the computations. The distribution has been calculated down to balls of ¾-in., but a sufficient amount of the smallest economical size of balls used in making up the charge should be added to yield a total weight of 100 per cent.

When balls of more than one size are regularly fed to the mill, it is assumed that the rate of wear is not changed; and the charge can be divided into different portions according to the ratio of balls of each size added. The composition of each portion is computed separately and the results are combined to obtain the size distribution in the mill. For example, if the ball feed consists of one-third 5-in. balls and two-thirds 4-in. balls by weight, the equilibrium charge for a 5-in. ball and twice the equilibrium charge for a 4-in. ball are added. The sum is divided by 3 to obtain the combined charge.

A wide range of size distributions can be obtained in this manner. If a certain equilibrium distribution is desired for an ore feed of a specific screen analysis, the proper ratio of ball-feed sizes can be computed from Table 1, according to the method given later in this paper.

TABLE 1.—*Calculated Ball Charges*

Ball Size	5″	4½″	4″	3½″	3″	2½″	2″	1½″	1″	¾″
n	61.95	61.65	61.30	60.95	60.50	59.95	59.30	58.45	57.30	56.47
N_F	62.45	62.15	61.80	61.45	61.00	60.45	59.80	58.95	57.80	56.97
P_F	118.8	118.9	119.0	119.3	119.7	120.6	122.3	126.5	141.1	171.7
Log P_F	2 075	2.0752	2.076	2.077	2.078	2.081	2.087	2.102	2.149	2 235
Grams per ball	8,400	6,120	4,296	2,881	1,816	1,049	537.3	226.8	67.20	28.32
200-mesh contact area, sq. mm	14.76	13.28	11.80	10.33	8.86	7.38	5.92	4.43	2.95	2.21
For 1000 pounds:										
Number of balls	54.0	74.2	105.5	157.4	249.6	432.2	844.0	2,000	6,750	16,000
Square meters surface	2.74	3.04	3.42	3.91	4.56	5.46	6.84	9.12	13.68	18.25
Square meters equilibrium surface	4.68	5.09	5.78	6.49	7.47	8.90	10.78	13.85	18.75	22.90
EQUILIBRIUM CHARGE, PER CENT CUMULATIVE ON SIZE										
No. 63	0	0								
No. 62	31.72	13.41	0	0						
No. 61	69.32	61.05	48.32	31.72	0	0				
No. 60	86.26	82.57	76.93	69.50	56.28	31.83	0			
No. 59	93.93	92.31	89.81	86.53	80.70	69.93	48.88	0		
No. 58	97 40	96.71	95.63	94.23	91.74	87.11	78.11	55.44	0	
No. 57	98.97	98.69	98.27	97.72	96.73	94.89	91.33	82.36	51.03	0
No. 56	99.68	99 59	99.46	99.29	98.98	98.41	97.30	94.50	84.75	64.10
No. 55	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00
INITIAL CHARGE, PER CENT WEIGHT										
5 in.	11.0	0								
4½ in.	21.2	12.0	0							
4 in.	16.8	21.5	16.3	0						
3½ in.	13.5	18.5	24.2	15.9	0					
3 in.	12.2	16.0	18.0	28.7	19.0	0				
2½ in.	9.6	12.2	15.8	21.2	31.2	21.0	0			
2 in.	7.0	8.8	10.9	14.9	21.9	35.7	25.5	0		
1½ in.	4.6	5.8	8.7	10.5	15.3	23.7	41.0	31.5	0	
1 in.	2.5	3.3	4.0	5.3	7.5	11.9	20.3	41.5	25.9	0
¾ in.	1.6	1.9	2.1	3.5	5.1	7.7	13.2	22.0	74.1	100.0
Total	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0

If the mill is fed continuously with balls of the selected sizes, and in the selected weight ratio, the size distribution of the charge should soon reach the indicated equilibrium value. This makes it possible to ration the ball sizes to the feed to suit the ore and grinding conditions, with the assurance that the desired size distribution will continue constant as the balls wear. This is predicated on the use of balls that have a relatively homogeneous composition.

Surface Area

The total surface area of the equilibrium ball charge, or the total grinding-surface area, excepting the mill lining, is found from the following formula, in which S equals the surface area in square meters per 100 c.c. of metal in the ball charge:

$$S = \frac{157{,}800 P_F}{2^{\frac{1}{2}(N_F-1)}}\{0.7994 - \text{antilog}\,[(0.1945)(55 - N_F)]\} \quad [6]$$

The computed value of S multiplied by 579 gives the square meters of total grinding-surface area of the balls per 1000 lb. of equilibrium ball charge.

The ball-surface area, in square meters per 1000 lb. of equilibrium ball charge, for each of the common sizes of balls fed to the mill is listed in Table 1. When mixtures of balls of different sizes are fed in a particular ratio, the resulting surface area is readily computed from the known composition of the mixture.

Ball Consumption

It can be shown from eq. 2 that

$$\text{Log grams weight loss per ton of balls} = K_2 - 0.107n \quad [7]$$

where K_2 is a constant for the production of a specified amount of new surface area and n is the ball-size number. According to this equation, the introduction of a larger ball into the mill charge will result in a decrease in the amount of ball weight lost by grinding.

Since tests have shown that under ordinary grinding conditions a change in the ball size does not affect the new surface area produced per ton of balls, but does change the size distribution,[1] it follows that an increase in the ball size results in a lower ball consumption per ton of material ground, or per unit of new surface area produced.

If the ball size is doubled, the ball consumption per ton ground should be reduced approximately 39 per cent; and with the same amount of new surface area produced as before, but with more very coarse and more very fine material in the product. Increasing the size number by two units is equivalent to doubling the ball size.

When the sole object of grinding is to produce new surface area, regardless of size distribution, an increase in the ball size might be advantageous; but when the object is to pass a certain mesh, or to unlock or expose certain minerals, balls larger than those necessary to break the ore may prove to be inefficient.

Ball-size Differential

The immediate object of most grinding installations, except perhaps in the cement industry, is to grind to pass a specified mesh. The most efficient conditions are those in which the largest amount of undersize particles is produced; not those in which the largest amount of new surface area is made. The production of particles much finer than the desired limiting size involves wasted energy input, and may be detrimental to the subsequent metallurgical processes. It is evident that the most favorable conditions are those in which a large amount of material just under the limiting size is produced, with a minimum of oversize and few very fine particles.

The best conditions for obtaining a product of this character are: (1) the use of a circulating load, (2) a short detention time in the mill, (3) a proper adjustment of ball sizes to suit the screen analysis and

hardness of the mill feed and the mill diameter and speed. The ball charge should be so selected that the largest balls present will efficiently break the largest particles in the feed, and the ball sizes theoretically should be distributed to correspond to the size distribution of the material fed.

The size and hardness of the mill feed are more important in the selection of the proper ball size than is the size of the product, since balls that are too large always produce wasteful undersize, and balls that are too small cause wasted contacts. Also, the use of a large circulating load greatly reduces the difference in average particle size between the feed and discharge ends of the mill. The most efficient grinding to a limiting size would be obtained by repeated contacts with balls of a size just sufficient to break the particles efficiently, and with only one particle broken per contact.

When there is an even distribution of small particles of the same size over the ball surfaces, a ball 2 in. in diameter will strike an individual particle ¼ as hard as a 4-in. ball in a mill of the same diameter, and a ton of 2-in. balls will strike four times as many particles per revolution of the mill as a ton of 4-in. balls. If the particles are large enough so that only one particle is struck for each ball contact, the 2-in. ball will strike each particle only ⅛ as hard as the 4-in. ball, but will strike a unit area of the particle ¼ as hard, because of the decreased area contacted. If several particles are broken per contact, it is obvious that any change in the number contacted will cause either overgrinding or waste of contacts.

When the diameter of an ore particle is doubled, which is equivalent to an increase of two units in the size number, the minimum amount of new surface area produced by failure of the particle under compression is increased four times. Consequently, an increase in the size number of the material to be ground requires an increase of an equal number of units in the minimum size number of the balls necessary to break the ore.

For any specified set of grinding conditions there is a minimum difference in the size number of an ore particle and that of the ball necessary to break it efficiently. Theoretically this difference should be constant for particles of all sizes in the mill, provided all of the particles are of the same hardness. It is designated as B, and is called the optimum ball-size differential.

Its numerical value is the difference between the particle-size number and the ball-size number that will break the particle most efficiently under certain specified grinding conditions. B decreases with an increase of mill diameter or of mill speed, since these result in an increase in the impact of each striking ball of the same size. The optimum ball diameter can be calculated directly from

$$\text{Ball diameter} = 2^{B/2}\,(\text{particle diameter}) \qquad [8]$$

If the proper ball diameter for any particle diameter is known, the value of B can be calculated from

$$B = 6.645 \log \left\{ \frac{\text{Ball diameter}}{\text{Particle diameter}} \right\} \qquad [9]$$

The value of B for an ore of medium hardness, ground wet in a mill of 6-ft. dia., is approximately four units, with about five units required for a hard ore. B equals 7 for an ore of medium hardness ground in a laboratory mill of 18-in. dia. and it is reduced to 8 in a mill of 12-in. dia. with a medium ore, and 9 with a hard ore. These values are tentative only, and subject to revision.

Contact Area

Spheres theoretically make a point contact, but the area within which two balls in contact will nip or crush a particle of a certain definite size can be readily found.

This area is $(\frac{1}{2})\pi dp$, where d is the ball diameter and p is the particle diameter.

The area within which a 200-mesh particle will be nipped, or the 200-mesh contact area, is equal to 0.01162 d sq. cm., where d is the ball diameter in centimeters. The contact area is directly proportional to the ball diameter, and also to the particle diameter. In terms of size numbers, the contact area between two balls of the same diameter doubles for an increase of two units in either the ball diameter or the particle diameter.

An increase of two units in the ball-size number decreases the number of balls per ton, and the number of contacts per ton to $\frac{1}{8}$ of its former value, and decreases the total contact area per ton to $\frac{1}{4}$.

The contact area for balls of different diameters in contact is somewhat more than that calculated for the smaller ball. When one ball is double the diameter of the other, the contact area is approximately 50 per cent more than that calculated for two of the smaller balls in contact; and it is 100 per cent more for a ball in contact with a flat surface, or with the mill lining.

The 200-mesh contact area, in square millimeters, for a single point of contact between two similar balls is listed in Table 1. The number of balls in 1000 lb.; the surface area in square meters of 1000 lb. of balls; and the surface area in square meters of 1000 lb. of the equilibrium ball charge in a mill fed with balls of different sizes, are also listed.

A decrease of two units in the size number of the ball decreases the ball pressure four times, measured in grams per square centimeter of 200-mesh contact area. It increases the total contact area per ton of balls four times, and doubles the total surface area of the balls.

These relationships are not limited to 200-mesh, but remain constant for any particle size. The energy of impact of a striking ball is directly proportional to the ball weight.

When the relationship between the ball size and the particle size is that of the optimum size differential B, the area between the two balls in contact within which the particle will be nipped or crushed is found from:

$$\text{Optimum contact area} = \frac{\frac{1}{2}\pi d^2}{2^{B/2}} \quad [10]$$

When a particle is nipped between two balls of the same diameter in contact with each other, and the ball-size number is B units larger than the particle-size number, the angle A by which the particle is nipped between the balls is found from

$$\text{Cos angle of nip} = \cos A = \frac{1}{1 + \frac{1}{2^{B/2}}} \quad [11]$$

The values of A for corresponding values of B are as follows:

B	A
4	37.00°
5	31.75°
6	27.25°
7	22.75°
8	19.67°

It is evident that in contacts where the ball-particle size differential is smaller than about five units, the included angle is too great for the particles to be actually nipped; hence comminution is necessarily accomplished in these contacts by impact rather than by attrition.

The void space between the balls in a mill charge is approximately 40 per cent of the total volume occupied by the ball charge. A study of the various crystallographic patterns in which balls of the same diameter can be arranged disclosed one form with a void space of 39.56 per cent. This is commonly called the cubical-hexagonal arrangement, and actually belongs to the pyritohedral class of the isometric crystallographic division. In this particular arrangement each ball makes eight contacts with surrounding balls. It can be assumed from these observations

that in a stationary mill charge where the balls are of the same size each ball makes eight contacts with its neighbors, and that the ball of average diameter in a charge of mixed sizes makes eight contacts.

Rationing of Ball Sizes

After the optimum ball-size differential has been determined, it might appear that the calculation of the ball-size distribution to give the maximum efficiency in grinding to pass a certain mesh size would be relatively simple, but a careful consideration of the factors involved shows that it is otherwise.

Certain assumptions are necessary if numerical results are to be obtained, and the validity of these assumptions may be questioned. The method described in the following paragraphs represents the results of repeated attempts to arrive at a satisfactory formula for ball rationing by logical processes, but until commercial tests are made it should be regarded as only a tentative solution of the problem.

If the charge contains balls that are too large, energy is wasted in overgrinding at each contact, and the total number of contacts is reduced. If the balls are too small to break the particles, their contacts are wasted, except for some slight reduction by attrition on the corners and edges.

The desirable conditions include the maximum possible number of useful, or breaking contacts, and the production of broken particles with the maximum possible diameters. The removal of projecting corners or edges by attrition is not considered in the following computations.

Inasmuch as we are dealing with a mixture of balls and particles, each including a wide range of sizes, and since contacts between all sizes are possible, the determination of the optimum ball-size distribution necessitates a study of all the probabilities.

If the feed particle size remains constant and the ball size is increased by one unit, the most probable particle diameter after a breaking impact decreases by $\sqrt{2}$. If the ball size is increased by two units the particle diameter of the product decreases by one-half.

This conclusion is based on the premise that the total surface production remains constant as the ball size increases, and the corollary that the amount of new surface area produced by each unit of contact area increases in proportion to the decrease in the total contact area.

The new surface area produced increases in the same ratio as the particle diameter decreases, and the net energy expenditure varies directly as the new surface area.

If the ball size remains constant and the feed particle size is reduced by one unit, the most probable product particle size is reduced by $1 + \sqrt{2}$, or by 2.4142 size units.

A decrease of one unit in feed particle size, with a constant ball size, doubles the number of probable ball-particle contacts, since the total contact area remains constant.

When the two preceding statements are combined, it is found that if the ball size remains constant and the particle size is decreased by one unit the efficiency of the useful contacts is reduced by $2.4142 - 2$, or by 41.42 per cent.

A comparison of the efficiency losses caused by overgrinding in useful contacts shows that a decrease of one unit in particle size increases the loss to 141.42 per cent.

This does not include the loss due to wasted contacts of undersize balls. The most desirable condition is obtained by adjusting the ball distribution in such a manner that the efficiency will continue constant for each ball size, so that it will remain equal to that obtained on the largest particle size.

Each decrease of one particle-size unit decreases the number of wasted contacts in the mixed charge by one-half, and 141.42/2 subtracted from 100 gives 29.29 per cent as the amount of adjustment

necessary in the ball distribution for each successively lower particle size.

The per cent cumulative weight retained in the ball-size distribution should be increased by 29.29 per cent over that retained in the corresponding size of the feed material entering the mill, for each successively lower size number, as explained below.

The per cent cumulative weight of the feed retained on the largest size number is equal to the per cent cumulative weight of the desired ball charge retained on that same size number increased by the ball-size differential B.

The per cent cumulative weight of the feed retained on the second size number is multiplied by 1.2929 to find the per cent cumulative in the desired ball charge retained on that size number plus B, that retained on the third size number is multiplied by $(1.2929)^2$, that retained on the fourth by $(1.2929)^3$; and so on. The smallest ball-size number calculated is that where the calculated per cent cumulative ball weight retained equals or exceeds 100 per cent.

The size distribution of the equilibrium ball charge calculated in this manner should theoretically give the most efficient results on the basis of grinding to pass a certain specified mesh size. It is still necessary to determine the composition of the *continuous ball feed* that will most nearly produce the desired equilibrium charge in the mill, and the size distribution of the *initial ball charge* that will most nearly approximate the desired equilibrium charge.

The suitable initial ball charge is readily calculated from the plotted per cent cumulative weight of the desired charge, using the method described previously under the heading "Initial Ball Charge."

The size distribution of the continuous ball feed necessary to maintain the desired equilibrium ball charge in the mill is calculated as follows:

The per cent weight of the largest ball size to be used is found by dividing the per cent cumulative weight retained on the largest size number in the desired ball charge by the per cent cumulative weight retained on the same size number in the equilibrium charge of a mill fed only with balls of this largest size (Table 1).

The per cent weight of the largest ball size used is multiplied by the per cent cumulative weight retained on each size number in the equilibrium charge, as listed in Table 1, and the result compared with the equilibrium ball charge. Additional smaller balls must be added to make up deficiencies in the equilibrium charge produced by feeding the largest balls alone.

At the largest size for which any deficiency exists, the amount of balls of that size necessary to make up the deficiency is computed from Table 1, and the per cent cumulative size distribution is found for the charge resulting from the addition of these balls to the mill. The process is continued until the entire composition of the regular ball feed has been calculated.

The agreement between the calculated equilibrium distribution and the desired distribution is usually not exact, and it is considered preferable to add the full calculated amount of the largest balls necessary, even though the desired amounts of the sizes immediately below this may be exceeded. The presence of balls ranging down to size 55 in the equilibrium charge usually partly compensates for this excess, since 100 per cent cumulative of the desired calculated charge is usually retained on a larger size number.

Closed-circuit operation usually returns to the mill a large amount of material that is much finer than the new feed, and may make it advisable to include small balls in the mill feed, which would not be necessary in open circuit.

Example

Operating data from a small mill treating 100 tons daily of gold ore were selected to illustrate the calculations. The mill is

7 ft. in diameter by 5 ft. long, and runs at 20 r.p.m., with a ball load of 8 tons of 5 to 3-in. forged-steel balls. It operates in closed circuit with a classifier, with a circulating load of 100 per cent. The largest particles in the mill feed are approximately 1 in., and the classifier overflow is 71.3 per cent through 200 mesh. The circulating load returned from the classifier has 15.6 per cent cumulative on 20 mesh.

represents the per cent cumulative weight retained on the ball size numbers N. It is given in column 6.

The percentage of 5-in. balls to be used in the regular, or daily, ball feed, is found by dividing the desired percentage of the largest size (22.00) by the percentage retained on this size with a ball feed that is all 5 in., as listed in Table 1. As 22.00 divided by 31.72 equals 0.694, the regular

TABLE 2.—*Data on New Feed*

1	2	3	4	5	6	7	8	9	10	11
Mesh	Size, n	Feed, Per Cent Cumulative	N ($B = 4$)	Factor	Desired Ball Charge, Per Cent Cum.	5-in. Balls (69.4 Per Cent)	2-in. Balls (30.6 Per Cent)	Equilibrium Ball Charge Per Cent Cumulative	Initial Ball Charge: Size, In.	Initial Ball Charge: Per Cent Wt.
Inch										
1½....	59	0	63		0	0		0	5	7.2
1......	58	22.00	62	1.000	22.00	22.00		22.00	4½	13.5
¾....	57	29.50	61	1.2929	38.13	48.10		48.10	4	11.1
½....	56	35.24	60	1.672	58.92	59.85	0	59.85	3½	10.6
⅜....	55	38.25	59	2.161	82.70	65.15	14.95	80.10	3	8.8
Mesh										
3....	54	40.61	58	2.794	(113.50)	67.60	23.90	91.50	2½	8.0
4....	53	42.26	57	3.613		68.60	27.95	96.55	2	13.1
6....	52	43.47	56	4.671		69.00	29.75	98.75	1½	14.1
8....	51	46.75	55	6.039		69.40	30.60	100.00	1	13.6
10....	50	49.68	54	7.808						
14....	49	51.70	53	8.019						100.00
20....	48	53.71	52	13.056						
28....	47	56.37	51	16.872						
35....	46	60.29	50	21.813						
200....	41	93.09								

Since 5-in. balls (size number 62) are sufficiently large to break 1-in. ore particles (size 58) in this mill, the ball-size differential, or value of B, is four size units.

The screen analysis of the combined new feed and circulating load entering the mill was computed from the available operating data, and is listed in column 3 of Table 2. The equivalent ball size N, which is the particle size n plus B, is listed in column 4, and the factors used for finding the desired equilibrium ball charge are given in column 5. The factor for the largest size is unity, the second is 1.2929, the third is $(1.2929)^2$, and so on.

The size distribution of the desired equilibrium ball charge is the product of column 3, and the factors in column 5, and

ball feed should contain 69.4 per cent of 5-in. balls by weight. The equilibrium distribution for 5-in. balls given in Table 1 is multiplied by 0.694 to obtain the distribution for the 5-in. balls alone in column 7.

Comparison of column 7 with column 6 shows that the correct amount of size 62 balls is present, and that there is an excess of about 10 per cent of size 61 balls and 1 per cent of size 60 balls in the actual equilibrium charge. There is a deficiency of 17.5 per cent of size 59 balls, so that it is advisable to complete the regular ball feed with balls of approximately this size, which corresponds to 2 inches.

The equilibrium distribution of 2-in. balls, given in Table 1, is multiplied by 0.306, and the products are listed in column 8.

The actual equilibrium ball charge obtained by feeding 69.4 per cent of 5-in. balls and 30.6 per cent of 2-in. balls to the mill is given in column 9, which is obtained by adding columns 7 and 8.

The commercial ball sizes used in making up an initial ball charge that will have the approximate distribution of the equilibrium charge are listed in column 10, and the per cent weight of each ball size used is given in column 11. This was calculated from the plotted values of column 9, according to the method described under the heading "Initial Ball Charge."

The inclusion of small balls in the regular mill feed is necessary in this calculation because of the circulating load. If the mill were operating in open circuit the use of 5-in. balls alone would give a satisfactory distribution, but the return of a large amount of relatively fine classifier sands to the mill requires the addition of some small balls to obtain the maximum production of minus 200-mesh material.

No allowance has been made for possible variations in hardness of the different size fractions of the mill feed. However, the excess in the ball charge of size 61 should be sufficient to take care of any probable variations.

It is evident that the proportion of smaller balls in the equilibrium ball charge can be increased above the values given in Table 1 by any desired amount, by including small balls in the regular mill feed. However, the proportion cannot be reduced below the values in Table 1, except by increasing the size of the balls fed, or by periodically removing the smaller balls from the mill.

Conclusions

The calculations indicate that in most mills operating in closed circuit with fairly large particles in the new feed, the proportion of smaller balls in the charge should be higher than that obtained by feeding only balls of the size necessary to break the largest particles.

It is probable that the distribution should be computed so that there is no deficiency of the larger balls at any size, and the unavoidable excess of very small balls of sizes 56 and 57 in the equilibrium charge is balanced by an equivalent excess in the larger sizes, as was done in the example given. It should be remembered in this connection that the balls are rationed according to the size distribution of the particles entering the mill, and not according to the average particle sizes within the mill.

Summary

A formula for the relative amount of wear of balls of different sizes in a mixed ball charge has been derived from a series of experimental measurements. This formula shows that a film of metal of any specified thickness is worn from the large balls slightly faster than from the smaller balls in the same charge.

A method for calculating the size distribution of the equilibrium ball charge in a mill, fed continuously with balls of one size, has been derived from the known relative rates of ball wear. The equilibrium size distributions for the commonly used ball sizes have been calculated and tabulated in Table 1.

The size distribution of the initial ball charge used in starting up a new mill, which is approximately the same as that of the equilibrium ball charge reached after the mill has been in operation for some time, has been calculated and tabulated for mills fed with balls of the commonly used sizes.

The surface areas of the various equilibrium ball charges have been calculated.

It has been shown from theoretical considerations that the ball consumption for a unit production of new surface area should decrease as the ball sizes are increased, but that the size distribution

of the ground product will ordinarily be less favorable.

The numerical difference between the size number of a particle and that of the proper sized ball to break it most efficiently is discussed. This difference is called the optimum ball-size differential, and designated as *B*.

The contact areas within which a particle of any given size can be broken or nipped are calculated, and are tabulated for 200-mesh particles and balls of various sizes. The angles of nip are calculated for different values of *B*.

A study was made of the rationing of ball sizes in a mill charge to conform to the screen analysis of the material entering the mill, and a method for calculating the desired ball-size distribution in the mill has been developed.

The calculation of the proper ball ration to be fed regularly to the mill in order to produce the desired equilibrium charge is described. The composition of the initial ball charge that would approximate that of the desired charge is shown by calculation. An illustrative example of these calculations is given.

The suggestion is made that the grinding efficiencies of mills with a fairly large feed particle size, which are operated in closed circuit, may be increased by the regular inclusion of some smaller balls in the mill feed. In some cases, where only one size of balls has been fed, the mill efficiency might be increased by changing to a mixture of balls of a larger size and of a smaller size than those used previously.

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The author desires to express his appreciation for the constructive criticisms offered by several members of the Allis-Chalmers Mining Division, including a critical review of the manuscript by Mr. W. L. Maxson.

DISCUSSION

(*E. W. Engelmann presiding*)

E. W. Davis,* Minneapolis, Minn.—Mr. Bond's paper would have been more convincing if actual records had been presented showing the size distribution of residual ball charges of commercial grinding mills operating under different conditions.

The paper that I presented to the Institute 20 years ago, and to which Mr. Bond refers, was a mathematical analysis based on information secured by carefully sizing the ball charge removed from an 8-ft. by 22-in. Hardinge mill after the mill had operated for a year. The ball charge was maintained by adding 400 lb. of 2-in. balls daily, and during the year of operation the balls that were worn out were nearly 10 times the weight of the residual ball charge. This seemed to be quite definite proof that the ball charge had reached a state of equilibrium, and using this information, it was shown that the rate of wear of each ball was proportional to its weight or the cube of its diameter.

It is admitted that at the time my paper was written the Miami test was the only accurate information on the subject available. A. L. Blomfield, then general manager of the Golden Cycle Mining and Reduction Co., undertook to make a 30-day test on a 6 by 6-ft. dry grinding mill. Through some misunderstanding with the mill operator, balls were not added for several days at one period, and then the charge was built up to normal by the addition of an abnormally large charge. This produced a nonconformity in the size distribution of the residual ball charge, as shown in the sizing test. The data from these two tests were the only information available when my paper was written. There may have been something radically wrong with these data, but at least the formula derived was based on results secured from full-size commercial mills.

Mr. Bond's mathematical analysis is based apparently on the results secured from a small laboratory mill, which he says indicates (no data presented) that ball wear is proportional to the square of the diameter of the ball. If this assumption is correct, Mr. Bond's conclusions are correct, and the data that I had

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to work with (which were presented) were in gross error. With all the ball mills that Allis-Chalmers have built and put into operation, it would seem that detailed results from commercial mills could have been accumulated before this paper was published, and the results incorporated in the paper. The paper certainly would have been much more valuable and much more convincing. As it stands, there is another Kick vs. Rittinger contention, this time over ball wear.

There is no possible excuse for a protracted argument, however, because it is such a simple matter to answer the question definitely by sizing ball charges removed from a dozen or so mills when they are stopped for relining. This can be done by rolling the balls down a long, slightly inclined slot, formed by placing two 20-ft. lengths of steel together at one end and 4 in. apart at the other. By placing partitions under the slot, the ball charge can be divided into as many increments as desired.

There must be considerable information available on this subject at the present time. If mill operators will report their results, the question can be quickly settled. Is ball wear proportional to the area or volume of the ball, or to some other function?

F. C. Bond (author's reply).—The experimental data on which the paper was based indicate that ball wear is not directly proportional to either the square or the cube of the ball diameter, but is intermediate between the two.

If the work done in grinding is the result of impact, the rate of ball wear would conceivably be proportional to the weight, or cube of the diameter, of the striking ball. In this case Kick's law would apply, and the coefficient of n in Eq. 2 would be the logarithm of 3, or 0.477.

However, if the grinding is done by attrition, or by rubbing and rolling of individual balls, the rate of ball wear should be proportional to the surface area, or to the diameter squared. Rittinger's law would apply, and the coefficient of n in Eq. 2 would be the logarithm of 2, or 0.301.

Many of the smaller balls in an equilibrium ball charge are not spherical, but have irregularly flat surfaces. This is an indication that a large part of the ball wear is the result of attrition, rather than impact.

The experimental coefficient of 0.345 indicates that the wear is proportional to the ball diameter to the power 2.21. This might be an expression of the relationship obtaining between impact and attrition grinding in the laboratory mill. If the relationship between impact and attrition grinding varies considerably in different installations, the coefficient of n may be appreciably different for each case, and the tabulation of equilibrium charges given in the article may not be of general application.

Variations in the surface hardness of the balls at different stages of wear would also have a considerable effect upon the size distribution. Forged balls ordinarily are harder at the surface than near the center; but such factors as composition and rate of cooling can greatly accentuate this difference, and cause a corresponding change in the slope of the distribution line. An increase in the difference between surface and interior hardness should increase the slope, and result in fewer small balls in the equilibrium charge. This is equivalent to an increase in the experimentally determined coefficient of 0.345, which may apply only to balls with a hardness variation similar to that of the balls used in testing.

There is certainly a need for the publication of more data on the size distribution of equilibrium ball charges by those in a position to obtain these data. The balls in each size fraction should be *counted* as well as weighed, and the weight of the average ball in each size group determined. Only in this way can corrections be made for the errors introduced by the irregular shapes of the balls, and by inaccuracies in the designated size limits.

Deleterious Coatings of the Media in Dry Ball Milling

By Fred C. Bond* and Fred T. Agthe,* Members A.I.M.E

(New York Meeting, February 1940)

When some materials are ground dry in a ball mill, a stage of comminution is reached at which the finely divided particles begin to adhere to the balls and to the mill lining. As grinding progresses, a coating accumulates upon the grinding media, which tends to cushion the impacts and thus retard reduction of the material, and finally becomes so thick that all grinding ceases. At this stage, the characteristic noise produced by the grinding media during normal mill operation changes to a dull, muffled sound.

In most instances, the formation of the coating marks the practical limit of size reduction. Industrial requirements for fine grinding frequently meet with this obstacle, and the tendency in many commercial operations to grind materials to a higher degree of fineness than formerly makes the solution of this problem an important one. The phenomenon is of far greater importance, therefore, than the limited amount of research and technical discussion devoted to it implies.

In order to learn more about the coating of different materials when ground dry to a high degree of fineness, certain laboratory tests were made from which it was found that different materials form coatings at widely different size distributions. It is probable that most materials will coat if they are ground fine enough; the exceptions have inherent lubricating qualities, like coal and graphite.

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In general, the softer materials tend to coat at coarser sizes than the harder or more brittle materials. Limestone, as an example, will coat more readily than quartz, even though limestones of different texture and composition exhibit great differences in coating potentialities.

Various materials were tested in the laboratory to show the variation in coating qualities. It was found that in most tests quartz must be ground to a fineness that would permit it to pass through 325 mesh before coating became markedly detrimental to the grinding operation. Siliceous rocks, however, tend to coat more readily than quartz. Granite, which is composed essentially of feldspar, quartz and mica, has been found to start coating at about 92 per cent through 200 mesh, and feldspathic minerals alone usually show a tendency to coat at the same fineness. Barite has been ground in the laboratory to a fineness of 99 per cent through 325 mesh without showing serious coating of the grinding media.

Gypsum coats readily. Apatite and phosphate rock usually tend to coat. Hematite usually coats, although when mixed with charcoal it does not do so.

In testing the carbonate rocks and minerals, it has been found that burned magnesite, dolomite, and dolomitic limestone coat, whereas calcite and calcined calcite do not coat so readily. It has been found that commercial glass, slate, talc, steatite and coke have little tendency to coat.

The relative tendencies toward coating during grinding operations, exemplified by

the laboratory grinding tests made of these materials, provide an interesting field for further investigation.

It is in the fine grinding of cement clinker, however, that coating becomes a major milling problem. The extreme fineness required of Portland and high-early-strength Portland cements involves problems of fine dry grinding, among which the coating of the grinding media is not of minor importance.

Records made of hundreds of clinker-grinding tests show clearly that clinker exhibits a wide variation in its tendency to coat. Some clinkers can be ground much finer than others before coating becomes a serious detriment to grinding. The reasons for the differences are not well understood, although it is observed that a cement clinker that has been cooled rapidly, or air-quenched when leaving the kiln, is more brittle and easier to grind than clinker that has been cooled slowly. The laboratory tests made thus far appear to indicate that clinker cooled quickly has a decreased tendency to coat.

Whether the decreased tendency toward coating caused by rapid cooling is due to the changed texture of the clinker in which the compounds have been fixed as definite crystalline structures, rendering the material more friable, or is independent of the crystalline structure, is a question that has not been settled definitely by laboratory determinations. There are persistent indications, however, from the studies made thus far, that clinkers that have been air-quenched have a decreased tendency to coat.

With the exception of the brittle materials, particularly quartz and commercial glass, the materials that definitely resist coating are those that have some lubricating or smoothing action on the grinding-ball surfaces.

The presence of a considerable amount of relatively coarse particles in the mill charge inhibits coating, which becomes serious only after these particles have been reduced in size. It is possible that coating would not be an important factor in grinding many materials that show a decided tendency toward coating if the charge contained 10 per cent or more of plus 35-mesh particles. These limits of quantity and mesh size should be considered only as an assumption based upon the limited extent of the experimental work and data accumulated from it.

It has been observed in the laboratory, and verified in the field, that the presence of a small amount of moisture in many materials tends to increase coating; but coating is not solely dependent upon the moisture content, because many bone-dry materials coat badly. The humidity of the atmosphere may have an influence upon materials that have a tendency to coat, especially the hygroscopic substances.

The temperature of the mill charge is not always a determining factor, although probably it has a definite effect, since coating may take place at room temperature, or even at temperatures well above the boiling point of water.

The character of the grinding media used appears to make little difference upon the amount of coating, and cast and forged balls produce identical results. Tests were made using smooth, hardened ball bearings as grinding media. The balls showed less coating while their surfaces remained smooth, but as soon as they became pitted and scratched from use, they coated as badly as forged or cast balls. Large and small balls in the same charge in the laboratory mill coat to about the same extent.

From these widely divergent laboratory results, it is not difficult to understand why there is no agreement as to the theoretical explanation of the cause of ball coating.

Hypotheses Related to the Causes of Coating

Static Electricity

Static electricity in the charge, and adsorption upon the particle surfaces, offer plausible explanations, and are frequently mentioned in this connection.

In regard to the electrostatic hypothesis as the cause of coating, it may be said that when unlike surfaces rub against each other friction induces an electrical potential between the two surfaces, and if the material is a nonconductor, an equivalent static charge accumulates on both surfaces. A surface charged with either positive or negative electricity is repelled by a similar surface carrying a similar charge, and attracted by a dissimilar surface carrying a charge of the opposite sign.

If the particles concerned are so small that the energy of the induced electrical potential is of the same order of intensity on each particle as that of the gravitational pull, the mutual repulsion of the particles, carrying a charge of the same positive or negative sign, causes physical manifestations that can be observed.

Friction between the particles and the balls in the process of grinding should produce an induced charge of one sign on the particles, and an equivalent charge of opposite sign on the balls and mill lining. The charge on the metallic surfaces would be dissipated immediately to the mill shell and ground, leaving the charge on the particles to cause a mutual repulsion and a consequent "fluffing up" of the material. To this extent the development of a static charge on the particles should tend to increase dispersion and decrease ball coating.

In a cement plant operating in the United States, the apparent specific gravity of the finely ground cement discharged from a grinding mill, which was operated in closed circuit with an air separator, was so low that standard cement bags could not hold the required weight of cement. However, with the addition of 0.5 per cent of water by weight to the mill feed, the apparent specific gravity became satisfactory, and normal packing was resumed. The added moisture increased the surface conductivity of the particles, and so decreased the amount of static charge carried; therefore, the mutual repulsion diminished.

It is difficult to imagine how coating could be increased by producing a mutually repellent charge on the particles forming the coating, especially when it is realized that after the balls become so completely coated that they can make no more metallic contacts there is no direct friction between the metallic ball surfaces and the particles.

Adsorption

Another explanation is that of adsorption, or changes in the surface character of the particles by the presence of dissimilar materials. This is undoubtedly an important factor, although the actual mechanism of the process is far from being understood.

Finely ground material cannot be packed tightly by shaking because of the air it contains, and it is not difficult to visualize this air as a partially adsorbed film surrounding the individual particles. However, apparently the presence of the air film would tend to prevent the particles from coalescing, just as the static electrical charge does, rather than to encourage the formation of coating.

Moisture in considerable amounts usually increases coating, although a small amount is beneficial in preventing coating of cement clinker, as noted previously. It is probable that the moistened or wetted surfaces of the particles tend to coalesce more readily, because of the removal of the surrounding air film. If this is true, the air film is an important factor in preventing coating, and the increase in

coating that is sometimes reported with a temperature increase in the mill charge may be explained as resulting from the decreased tenacity of the air film caused by increase of temperature.

Certain impurities or addition reagents may be adsorbed upon the particle surfaces, and if these have a strong affinity for air, or are wetted with difficulty, they may be of considerable help in preventing coating by the retention of the protective air film around the particles. Such materials as resin or coal, which occasionally are added to prevent coating, may be included in this class. However, the relatively immense surface area of finely ground material, and the amount of reagent necessary to coat this material, must always be considered.

The manufacturers of one reagent for prevention of the coating of cement clinker during grinding recommend that it be used in the amount of 0.067 per cent of the weight of the material. If the ground material has a specific surface of 1800 sq. cm. per gram, and the film thickness of the adsorbed reagent is 10 millimicrons, it may be shown by calculation that the reagent is sufficient to cover only 17 per cent of the surface area of the material. Since it is generally conceded that adsorption is stronger near the corners and edges of a particle than it is on the flat surfaces, this amount of surface alteration may be sufficient to alter the grinding characteristics considerably.

It appears, therefore, that the principal manner in which adsorption upon the particle surfaces may decrease coating is in the stronger retention of the air film upon the adsorbed surfaces.

Mechanical Hypothesis

A third explanation of the cause of ball coating is purely mechanical. The balls strike each other with considerable impact, and particles may be tamped and rammed together upon the uneven ball surfaces to such extent that a resistant coating accumulates, irrespective of the effects of static electricity and adsorption upon the particle surfaces.

This hypothesis is supported by the observation that ball coatings occur as a laminated concentric structure, which is increasingly dense and more consolidated as the metallic ball surface is approached.

Laboratory Tests

In view of the conflicting premises in these hypotheses, laboratory tests were undertaken. A sample of cement clinker that was known to coat easily was obtained. Minus 10-mesh samples of this clinker were ground dry with ⅞-in. forged-steel balls in a small porcelain jar mill. Each sample was ground for 30,0000 revolutions under constant conditions, except for the variables introduced to affect the coating, and the specific surface area of each product was determined with a Wagner turbidimeter. The results obtained are listed in Table 1.

Test I was a control test. The cement clinker was ground for 30,000 revolutions under standard laboratory conditions. The balls were completely coated, and the rate of grinding near the end of the test was evidently greatly reduced.

The coated balls were washed in gasoline, but the coating adhered so strongly that it could not be rubbed off completely between the fingers. A turbidimeter determination of the surface area was attempted on the coating removed by washing. The reading obtained was 3642 sq. cm. per gram, but this is probably too low, because the material was too fine for accurate turbidimeter results. The turbidimeter readings indicated that no particles larger than 20 microns in diameter were present, and a screen analysis of the ball coating showed 99.8 per cent through 325 mesh.

Particles of rock less than about ½ micron in diameter may remain suspended

indefinitely in water, and are said to be colloidal. They cannot be examined satisfactorily under a microscope because the wave length of light, which is about ½ micron, approaches too closely the diameter of the particle examined. The atoms are spaced in the order of 4 millimicrons, or 0.004 microns, apart.

ball could be magnified to the size of the earth, the scars formed upon them in grinding would appear as canyons ranging up to about two miles in depth and width, and the individual atoms would be spaced perhaps 10 ft. apart. Particles wedged into these scars would appear as mountain ranges.

TABLE 1.—*Results Obtained with Cement Clinker*

Test No.	Gypsum Added, Per Cent	Special Reagent	Product			Amount of Coating
			−200 Mesh	−325 Mesh	Specific Surface	
I	3	None	85.1	78.8	1715	Large; balls completely coated.
II	None	None	88.0	78.6	1525	Less than I.
III	None	3 % ZnO	84.6	76.8	1610	Worst of all.
IV	3	0.88 % H_2O	87.8	80.3	1800	Very little.
V[a]	3	0.75 % H_2O 0.13 % patented lignin compound	99.6	95.8	2330	Almost none, best of all tests.

[a] Twice the amount of water and lignin compound recommended previously.

In order to explore the mechanical hypothesis further, the washed and unwashed coated balls were examined under magnifications ranging up to 500 diameters, and the following observations were made:

The metal ball surfaces are very much roughened. They are completely covered by pits and scratches. The diameters of the pits and the width of the scratched channels are of the order of from 1 to 3 microns. Some of the roughened structures are larger than this, and many secondary structures are smaller, but this range is the most important.

The thin film of ball coating that remained locally attached to the balls after washing in gasoline consisted of extremely fine and closely packed particles. The particle size range observed was from 5 microns down to ½ micron. No particles larger than 5 microns were seen, and very few larger than 4 microns. They were so tightly tamped and rammed together that very little void space existed.

To visualize the relationship between particle size and the balls used to produce them, it may be said that if a small grinding

The structure and arrangement of the inner coating adhering to the balls indicated that the very fine particles had been forced into the scratches and pits, thereby covering the ball surfaces. They were wedged into place so tightly that they could be removed only with difficulty. The inner concentric coating contained only particles that were of the order of size of the pits and scratches on the balls.

Slightly larger particles were found immediately above the inner coating, which apparently had been wedged into place between the smaller particles beneath. The largest of these particles exceeded only slightly the 5-micron diameter limit of the inner coating.

As the coating became thicker, the particles were held more loosely, with more void space between them, and the average particle diameter increased slightly. The outer coating of the unwashed balls contained a few particles 15 microns in diameter, but none was seen above 20 microns. The outer particles were only loosely attached, and could be rubbed off easily between the fingers.

It should be remembered in this connection that the loose material that was not definitely coated on the balls contained 14.9 per cent of particles larger than 74 microns in diameter, as shown by the screen analysis of the product.

The ball coatings removed by washing with gasoline analyzed 0.81 per cent sulphur, or 50 per cent higher than the average charge placed in the mill. This shows that the gypsum tends to concentrate in the ball coatings. Since the gypsum is much softer than the clinker, and therefore is ground finer in the mill, its tendency to concentrate in the coatings is caused apparently by its smaller average particle size.

Test II was the same as the first test except that the 3 per cent gypsum was omitted. The balls were not coated as badly as in the first test, and the screen analysis showed a slightly finer product, but the specific surface of the product was less.

Gypsum probably is reduced to a fineness that permits all of it to pass through 325 mesh early in the grinding operation, so that screen-analysis results indicate improved grindability in the absence of gypsum, due to decreased ball coating. The low specific surface obtained is caused, therefore, by the absence of the extremely fine gypsum in the product.

The ball coatings removed by washing with gasoline screened 100 per cent through 200 mesh, and 90.3 per cent through 325 mesh. A turbidimeter determination of the coatings gave a specific surface of 2320 sq. cm. per gram. The average particle size of the ball coating is much smaller than that of the loose material in the mill, but it is larger than that of the first test.

Microscopic examination of the balls confirmed the observations of the first test, and indicated strongly that the decrease in coating was caused by the presence of fewer particles below 5 microns in diameter in the product.

Test III was made to confirm the observations of the first two tests.

If coating depends upon the number of fine particles present, the introduction of a very soft material, which breaks readily to form extremely fine particles, should increase the amount of ball coating greatly.

The test procedure was conducted in a manner similar to that of the first test, except that the gypsum was replaced by 3 per cent of zinc oxide. The zinc oxide consisted of agglomerates of particles only a few microns in diameter, and proved to be ideal for the purpose intended; since the ball coating was very much worse than in the previous tests, and the amount of grinding was markedly reduced. The inside of the mill was plastered with adhering material, and the ball surfaces were covered with a deep coating. The measured specific surface of the ground product was undoubtedly increased by the presence of extremely fine particles of zinc oxide.

The results of these three tests indicate strongly that ball coating is a selective process involving the extremely fine particles of the mill charge.

Test IV was made to determine the effect of adding a small amount of water to the charge.

It is known that usually the presence of moisture in the charge increases the tendency to coat, but that cement clinker appears to be an exception to this rule. The test procedure was the same as that of test I, except that 0.88 per cent of water by weight was added to the charge before grinding. The balls were only slightly coated, and more grinding was accomplished than in any of the previous tests.

The beneficial action of the water can be explained as an effect of the selective hydration of the very fine particles of cement. If these particles are primarily responsible for ball coating, their removal by agglomeration and rapid hydrolysis makes finer grinding possible.

Test V was the same as test IV except that 0.75 per cent of water and 0.13 per cent of a patented lignin compound were used instead of 0.88 per cent of water. The results were much better than those obtained in any of the previous tests. The balls were only very slightly coated after 30,000 revolutions, and the mill product was relatively fine. This test serves to indicate that the addition of small amounts of a specially selected reagent may greatly decrease ball coating in specific instances.

Discussion of Results

The results of these five tests are consistent, and seem to indicate strongly that ball coating is a selective process, starting with the smallest particles in the charge and gradually increasing with the depth of the coating. Occasional larger particles included in the coating may be considered as having been surrounded, or trapped.

According to this hypothesis, coating is largely a mechanical process. The ball surfaces are covered with pits and scratches formed during grinding, which range in width at the ball surface up to 5 microns. Particles below this diameter in the mill charge are continually packed and tamped into these openings during grinding by impacts with other balls. Often the particles are probably broken and forced into the openings by the same impact. Once the initial coating of the particles begins to adhere to the ball surfaces, other and larger particles are rammed into place among those which are first attached, and coating progresses rapidly. Finally it becomes so thick that the impacts are cushioned effectively, and grinding practically ceases.

The maximum diameter of the particles ranges from 5 microns at the inner coating, which is attached directly to the ball, to perhaps 20 microns or more at the outer coated surfaces. In all tests the coating has been found to be composed of the smallest particles in the mill charge. This mechanical hypothesis greatly reduces the importance of the theory that static electrical charges are a factor in promoting ball coating, and requires a somewhat different explanation of the effects of adsorption from those previously advanced.

We have seen that materials that tend to resist coating may be divided into two classes: brittle materials, and materials that have a lubricating or smoothing effect on the ball surfaces.

It is evident that relatively brittle materials, such as quartz or glass, do not tend to form extremely fine particles during grinding. The scarcity of the minus 5-micron particles necessary to start the coating process probably accounts for the behavior of these materials. When they are ground so fine that the mill charge includes the necessary quantity of minus 5-micron particles, they are observed to start coating, but much more grinding is necessary to reach this fineness than with a softer, or more friable material, or a material that contains a portion of relatively soft substances.

Adding 3 per cent of zinc oxide, or of gypsum, would undoubtedly greatly increase the coating of a mill grinding quartz, although limited time did not permit the assumption to be verified by test.

Granite coats more readily than quartz, because it contains some relatively soft substances, as well as some finely divided (colloidal) materials included between the larger crystal boundaries.

The materials that have a lubricating effect on the balls—for instance, graphite, coal, resin, and coke—resist coating for a different reason. They are soft, and produce an abundance of minus 5-micron particles during grinding, but these particles are effective lubricants, and permit larger or harder particles to slide free from all entrapments. There is probably some attraction, or adhesion, between these particles and iron, so that the uneven ball surfaces are coated with them, and harder or larger particles are not held in the open-

ings into which they have entered, but are allowed to drop out because of lowered frictional resistance.

The majority of the pits and scratches are unquestionably much larger at the top, or mouth, than at the bottom, and tend to have an irregular V-shaped cross section. Microscopic examination has shown that their depth is about the same or less than their width. Particles held in them are wedged into place between the sides of the openings, and the presence of any adhering lubricating material on the sides would tend to release the particles.

Little is known about the micromechanics of lubrication, and even less regarding the surface tension of solids, but some such explanation as that given above is necessary to satisfy the known conditions. The balls are lubricated by a film of finely divided adhering solids, which tend to fill up the scratches and pits, and to make the surfaces smooth. These particles are soft, and roll or slide upon each other so readily that harder or larger particles, the surfaces of which are also partly coated with the finely divided lubricants, are not held within the ball-surface openings. The effective coefficient of friction between the ball surfaces and the solids is therefore reduced.

After such material has been used in a mill, the balls are noticeably smooth, and even greasy to the touch, as contrasted with their rough feeling after grinding hard materials alone.

The difference between a soft material, or lubricant, like graphite, and a soft material that causes coating, like gypsum or zinc oxide, is probably related to texture, which is a function of its crystalline structure, and possibly also to its tendency to adhere to iron or steel.

Graphite has lubricating properties, because it has one crystal plane that offers very little resistance to shear; because it is soft rather than brittle, and probably because it tends to adhere to other solids. Coal and coke may exhibit similar properties in the finely divided particles formed by ball impacts, but coal may also release some hydrocarbons, which serve as lubricants. Certain wood products, like resin and probably lignin, have similar properties. For example, it is known that a small amount of bituminous coal, or resin, fed into a badly coated fine-grinding cement ball-peb mill, or tube mill, will reduce coating.

The effect of liquid lubricants is much the same as that of solids, except that probably they are held upon the ball surfaces by adsorption. Such films are thinner, and are perhaps mono-molecular. There is a substantial reduction in the amount of a liquid reagent required to inhibit coating. A specified amount of liquid reagent is ordinarily more effective than a similar amount of a solid reagent.

Adsorption of a liquid or gas upon a solid surface is a familiar phenomenon, and the adhesion of finely divided solid particles to a solid surface is certainly somewhat analogous, although little is known of the forces and surface tensions involved.

The authors believe that the adhesion of solids and the adsorption of liquids upon the ball surfaces is of more direct importance in preventing coating than is adsorption upon the particle surfaces of the material. Since coating begins with tiny particles being hammered into the roughened ball surfaces by impact, the prevention of coating depends primarily upon the condition of the ball surfaces, and any adhering films existing thereon that cause a lessening of the frictional resistance which holds the particle entrapped.

A lubricating film present on the ball surfaces is removed rapidly by transfer to the particles contacted, so that the lubricating reagent must be fed to the mill continuously if ball coating is to be permanently inhibited.

The interesting observation has been made that "Concavex" grinding media

show less tendency to coat on the concave, or cupped surfaces, than on the convex or spherical surfaces. A charge of Concavex used for fine grinding may have the convex surface badly coated, while the concave surface remains smooth and bright. However, it is probable that all of the surfaces will ultimately coat as comminution progresses.

The cause of this behavior is reasonably obvious. When an external force is applied to "Concavex" mediums in contact at a cupped surface, the resulting relative motion is that of sliding or rubbing; and particles reduced between the two surfaces are broken by shear. Any very fine particles wedged into the scratches tend to be wiped out and removed, and the surface is polished.

On the other hand, convex spherical surfaces in contact tend to roll upon each other, rather than to slide, and particles reduced by this contact are broken by compression. Such particles tend to be forced and wedged into the scratches on the balls, so that the surface becomes coated. The contacts made between the convex surfaces of the "Concavex" are of this nature.

The convex surfaces are thus subject both to rubbing, when they make contact within a cup, and to rolling compression when they contact each other; while nearly all of the contacts made by the cupped surfaces are rubbing contacts. The difference in the amount of coating produced by the two types of contact indicates clearly that sliding or rubbing reduce coating, while rolling or compression contacts tend to promote it.

Microscopic examination shows that the concave surfaces are marked with long scratches, while the convex surfaces are covered with shorter scratches and pits, similar to those on ball surfaces.

The differential coating observed on "Concavex" surfaces supports the mechanical hypothesis previously advanced.

Some suggestions as to the possible methods for combating ball coating, which will serve to improve fine grinding operations, can be made.

It has been noted that the presence of relatively large particles in the mill charge reduces coating, presumably by wiping off, or shearing the coating as it forms, or by penetrating and loosening it. In certain cases, it may be possible to operate fine-grinding mills which are in closed circuit with an air separator so that some very coarse material is discharged continuously and returned to the mill from the air separator oversize spout. Such particles should probably be larger than 35 mesh, and the rate of the return to the mill should be great enough to insure normal grinding operation made possible by cleaner grinding media.

An improvement might be obtained by including a very few large grinding balls in the mill charge. Contact with these balls should tend to shear the coating from the smaller balls; however, this suggestion should apply only to ball-peb mills or to tube mills using metallic grinding media for fine-grinding operations.

Another possibility pertinent to the reduction of ball coating concerns the composition of the balls themselves.

Balls that are harder than the substances they are used to grind must wear smoothly, so that surface roughening, which furnishes the footholds for incipient coating, may be reduced to the minimum consistent with low over-all unit grinding costs.

There has not been an insistent demand for high-quality grinding media, although there appears to be a growing tendency on the part of mill operators to recognize the importance of quality as a means of reducing grinding costs. It is possible that more consideration will be given to the subject in the future, particularly with regard to wearing properties that are conducive to smooth-surfacing.

As a means of reducing ball coating, the addition of a cheap solid lubricant, such as coal, may be used advantageously when its presence in the ground product is not detrimental. The use of liquid reagents, or solutions of solid reagents, appears to offer great possibilities, because of the relatively small amounts of these substances required to affect the ball surfaces.

Certain cheap oils or dissolved salts may be found highly beneficial in certain cases cases that may make their use economically feasible. The improvement obtained by adding a patented lignin compound in grinding cement clinker has been cited, and may be considered an indication of the latent possibilities in this direction.

The discovery of new reagents will require further careful and extensive research. This paper, therefore, should be considered only as a preliminary step toward investigations devoted to additional study of the entire problem. It is hoped that it may arouse interest and prove to be an aid in the orientation of future work.

Acknowledgments

The authors are indebted to Mr. Ewald R. Krueger, of the Crushing and Cement Laboratory of the Allis-Chalmers Manufacturing Co., for certain laboratory test results and many of the observations cited regarding the coating of specific materials.

DISCUSSION

(Charles E. Locke presiding)

O. C. Ralston,* College Park, Md.—While I do not wish to become a champion of an electrostatic theory, I wish to point out that I believe Mr. Bond has dismissed such a theory too hastily. The favorable effects of moisture addition in delaying adherence of coating would support an electrostatic theory because of the known effects of moist atmospheres in preventing generation of frictional electricity and in discharging any charged surfaces. Furthermore, electrostatic polarization of minerals whose crystal systems are not isometric is known. Heating by the mill could easily cause pyroelectric polarization of a material like quartz. A beaker full of quartz sand stirred with a glass rod on a hot plate will reach a temperature at which the sand sticks together in clumps, as though it had been wetted. Each quartz grain becomes a dipole with one end positively charged and the other negatively charged. Similar polarization can take place in quartz by the piezoelectric effect due to strain of the quartz crystals. Such dipoles are not easily discharged and have been known to retain their polarization, analogous to little permanent magnets, for considerable periods of time. Hence either frictional electric charging or polarization due to pyroelectric or piezoelectric phenomena might easily be present. Some of the materials ground in such a dry mill may form coatings due to these forces. Of course, minerals that crystallize in the isometric system should not do this.

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F. C. Bond and F. T. Agthe (author's reply).—Moisture tends to decrease the coating only in the case of cement clinker, and this presumably results from the selective hydration of the extremely fine particles. As far as is known, the presence of moisture promotes the formation of coating with all other materials.

As the authors are able to visualize the action within a ball mill, it seems that the continual pounding of the material by the balls constitutes a force of far greater magnitude than the mutual attraction or repulsion of the particles caused by their electrostatic charges. It appears that a particle would be forced into a crevice by the impact of a ball, regardless of whether it carried an electrostatic charge of the same sign or of opposite sign to that of the particles it is penetrating.

This may be an oversimplification of the processes involved. However, any comparison of the strongly adhering ball coating formed in a commercial mill with the extremely weak adhesion resulting from heating powdered quartz must tend to confirm the importance of the mechanical hypothesis.

Ball-mill Liners

By Warren L. Howes,* Member A.I.M.E

(New York Meeting, February 1943)

This paper deals primarily with an investigation of ball-mill liners that was conducted by the writer over a period of six years at the Mammoth mill in Arizona. The investigation covered a wide variety of designs of liners in both primary and secondary mills. The comparative results are presented herein as evidence of the extent to which design influenced liner life and mill performance.

Flowsheet

The feed to the Mammoth mill was a complex, oxidized, gold-silver-lead-molybdenum-vanadium ore with rhyolite, andesite and granite as gangue. Although the ore varied between stopes, the long-term average in character remained substantially the same. Capacity was 560 tons per day.† Fig. 1 shows an abbreviated flowsheet of the grinding section of the mill.

Primary Ball Mill

A 3-ft. Symons shorthead crusher in open circuit crushed to ¼-in. maximum one-way dimension the feed to the 5 by 10-ft. Marcy open-end ball mill. The mill rotated at 80 per cent of critical speed and was charged with 4-in. forged steel balls. Cast balls were used for a few months at a slight reduction in ball cost, but breakage of the large balls reduced the ball size and grinding capacity to a point where the over-all costs suffered out of proportion to the saving in cost of balls.

For metallurgical reasons, the finished product of the mill was maintained at minus 8-mesh. This coarse product, plus the very abrasive ore and the absence of fines in the feed, caused abnormally rapid wear of the mill liners. The resultant high operating cost led to the most extensive phase of the investigation.

Table 1.—*Primary Mill-shell Manganese-steel Liners*

Type	Number of Sets	Average Weight, Lb.	Average Number Operating Days	Average Weight per Day, Lb.	Percentage of Original Cost
Block, solid	1	20,331	92	221	100
Block, slotted	1	18,510	104	178	81
Undulating, slotted....	1	20,908	63	332	150
"Special," slotted undulating block.....	1	26,000 (est.)	115	226	102
Double-step, slotted, gusseted..	1	18,606	120	155	70
Double-step, slotted....	5	17,950	128	140	63
Double-step, solid......	1	19,119	210	91	41

In order to clarify the difference in design of the various types of liners used, Fig. 2 shows the profiles with the arrangement in which grinding balls of maximum size would lie on the face of the liners.

The results with each type of shell liner in the primary mill are shown in Table 1. Since most of the testing was done with

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† Details of the Mammoth operation appeared in the *Engineering and Mining Journal* in December 1941.

liners made of manganese steel, only the performance of the liners made of this material is shown, in order to limit the comparison to one variable—the design.

thick at the lifter and 2¼ in. thick at the valley. When worn out, each lifter was narrowed to a knife-edge but was still 3 in. thick, and the valley of 2¼ in. was worn

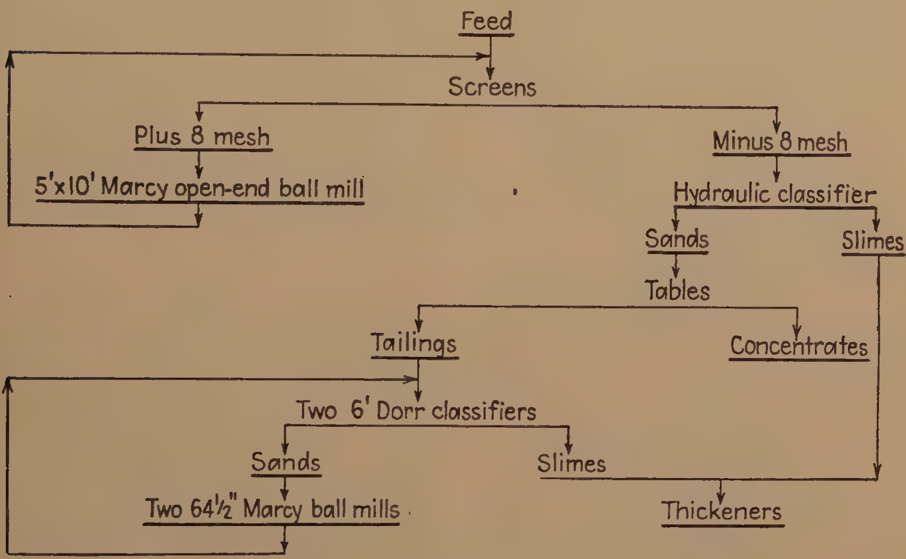

FIG. 1.—ABBREVIATED FLOWSHEET OF GRINDING SECTION.

through completely. In other words, 1¼ in. of metal was wearing from the top of the lifter while 2¼ in. was wearing from the valley. This indicated that the lifters were either too high or too far apart, or both.

During the first three weeks of operation with the block-type liners, the grinding capacity was abnormally low. This was caused by the abrupt leading wall of each lifter, which picked up large masses of balls and threw them across the mill, giving the effect of a mill run at an excessive speed. After the leading wall of the lifters had been beveled by natural wear, the grinding capacity improved appreciably.

BLOCK-TYPE LINERS

The initial shell liners used in the primary mill were of the conventional solid block type without slots. These lasted 92 days.

Because of previous good experience with so-called "slotted" liners, the next set installed in the mill was the same block type with slots in the face. Three broken rows of slots were arranged longitudinally in each liner segment, one down the center of the lifter and one in the valley on each side of the lifter. The slots measured 1¼ in. wide and 12 in. long and extended to within 1¼ in. of the back of the liners. A 2-in. web, or solid area, was left between the ends of the slots. These slots reduced the weight of the liners by about 9 per cent and the life was 104 days. The ostensible reason for using these slots was to improve the heat-treatment and uniformity of the steel.

Liners of the block type were 4¼ in.

UNDULATING LINERS

The high rate of wear and the loss in grinding capacity with the block-type liners led to an experiment with an undulating type. Each section of these liners carried two low lifters and five rows of slots and

was 4¼ in. thick at the lifter and 3¾ in. at the valley.

Although these liners were the heaviest of any used, they lasted the shortest length of time. The first set wore out in 63 days, and the manner of wear was interesting (and appalling) to watch. With these liners the grinding capacity was satisfactory for the first four weeks of operation until the lifters had disappeared. Then, excessive slippage took place, causing the formation of multiple grooves, after the fashion of ball races, in the faces of the liners. The rate of wear increased tremendously, as evidenced by the wearing out of a thickness of about 3¼ in. of manganese steel in five weeks of operation. In addition to this high rate of wear after the lifters had disappeared, the grinding capacity was reduced about 15 per cent by excessive slippage. Further evidence of excessive slippage was the very irregular consumption of power, as indicated by the ammeter connected to the mill motor.

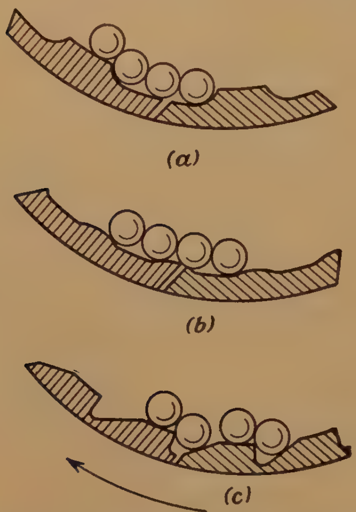

FIG. 2.—PROFILES OF VARIOUS PRIMARY MILL-SHELL LINERS WITH ARRANGEMENT OF BALLS OF MAXIMUM SIZE.

a, block type; *b*, undulating type; *c*, double step.

Because another set of the undulating type of liners had been ordered, they were used for 33 days, until the lifters were worn off. The liners were then removed and special manganese-steel lifters were plug-welded to the smooth faces of the liners, making them in effect block-type liners. These liners lasted 83 days. The welded lifters stayed in place fairly well, only two of the original 20 bars working loose during the run. The combined runs with this set of liners are consolidated under the heading of the "special" liners in Table 1.

DOUBLE-STEP LINERS

The foregoing experiences made it evident that there are critical relationships between the lifter and valley widths and depths. The possible role of ball size in liner design was also considered. The attempt to rationalize these factors led to the initial design of double-step liners.* The width of the lifters was one ball diameter and that of the valleys slightly more. The valley walls formed an angle of about 55° and permitted the seating of a 4-in. ball nearly to its center.

With these proportions of lifters and valleys, each liner section carried two lifters. In order to avoid wear at the joint between sections, the joint was made at the bottoms of alternate valleys. This placed a valley running down the center of each segment, along which breakage was possible. In order to strengthen this possible weak point, two bridges or gussets 5½ in. wide, and flush with the tops of the lifters, were cast integrally across the valley in each liner section. During the ensuing wear, about ⅜ in. more steel was worn from the liners in a circular strip around the mill covering these gussets than was worn on

* As finally evolved, the term "double-step" is not fully descriptive of liners designed according to the size of balls used in the charge. This name was given the early designs because each liner section happened to carry two lifters instead of the conventional one lifter per section. Subsequent designs have varied from 1½ to 3 lifters per section, dependent upon bolt intervals and ball sizes.

either side. In retrospect, it is apparent that the flat area three ball-diameters long, across which balls could slide before coming to rest in a valley, caused excessive wear, amounting to about 10 per cent, despite the presence of keyed balls in the valleys on either side of the gussets and at the joints fore and aft.

The grinding capacity with the double-step liners was excellent at the beginning of service and this capacity was maintained throughout the life of the liners. The reduced weight and increased liner life, to 120 days, also represented an appreciable reduction in liner cost.

Because of the excessive wear in the region of the gussets, these were omitted from the next set of double-step liners. No breakage along the central valley in each section occurred with these liners, even in materials subsequently tried, which were considerably more brittle than manganese steel. There was an appreciable reduction in weight and the average life of five sets of this type of double-step liner was 128 days, the best performance of any type up to this time.

In view of the beneficial experience with slots in block and shiplap liners, all of these double-step liners carried a row of slots down the center of each lifter. However, during the life of the liners there was a marked tendency by the webs, or solid areas on the lifters between the slots, toward standing out, indicating less wear where the lifters were solid. A set of solid double-step liners was tried in the same pattern but without the slots, and showed a very substantial improvement in the life of the liners, to 210 days. Apparently the slots in the double-step liners were unnecessary from the standpoint of heat-treatment, and they weakened the walls of the lifters so that peening and spalling from ball action shortened the life of the liners appreciably.

The performance of the solid double-step liners in the primary mill represented a reduction of 59 per cent from the consumption at the beginning of the investigation.

End Liners

Records of the investigation of end liners in the primary mill are shown in Table 2.

Table 2.—*End Liners in Primary Mill*

Type	Number of Sets	Average Weight, Lb.	Average Number of Operating Days	Average Weight per Day, Lb.	Percentage of Original Cost
Smooth.....	3	1,800 (est.)	80	22.5	100
Modified ribbed....	2	1,980	146	13.6	60
Multiridged.	1	1,730	142	12.2	54
Double-step, slotted....	1	1,955	209	9.4	42

The end liners first used in the primary mill were the conventional smooth type with some compensation for difference in wear over the radius of the mill. Three sets of these liners were worn out, with an average life of only 80 days per set.

The first revised design was a so-called modified rib type. The liner thickness was increased in the area of most severe wear and 12 ribs, 2½ in. wide and 1½ in. high, were superimposed on the face of the set. This led to a substantial increase in the life of the end liners, to 146 days.

The next type of end liner used was a light multiribbed design with sharp-shouldered ribs radiating to meet the lifters on double-step shell liners. This type of liner gave still lower liner consumption.

Ultimately put to use was a full-bodied step-type end liner with V-section valleys and full-width lifters radiating to meet corresponding lifters on double-step shell liners, resulting in a liner life of 240 days and the lowest liner consumption.

In this series of tests the consumption of end liners was reduced 58 per cent from the original consumption of smooth-end liners.

Secondary Ball-mill Circuit

The secondary mill circuit consisted of two 64½ Marcy ball mills, operating at about 80 per cent of critical speed, in closed circuit with 6-ft. Dorr classifiers. The minus 8-mesh feed was ground to 90 per cent minus 65-mesh.

The ball charges were maintained either by a straight addition of 2½-in. forged steel balls or a rationed charge of two thirds 2½-in. and one third 2-in. cast balls, depending on market conditions. The rationed charge of cast balls was needed to maintain full grinding capacity through proper balance of ball sizes. The large cast balls had a correspondingly large, soft and porous core which broke up after the balls had worn down to about 1¼-in. diameter, thus leaving the charge deficient in small balls. The addition of smaller cast balls, with correspondingly smaller cores, made up the deficiency and enabled full grinding capacity.

The profiles of the various types of liners used in the secondary mills are shown in Fig. 4, and the results with manganese-steel liners are summarized in Table 3.

The liners first used in this circuit were of the solid block type. As in the primary mill, these wore out selectively at the joints in the valleys, incurred high scrap loss and did not grind well when new. A series of tests followed with various shiplap liners in solid, slotted and spiral designs. These liners gave more uniform grinding capacity and the slotted and spiral designs showed a slight reduction in cost of liner.

Fig. 3.—Slotted double-step liners for primary mill with arrangement of 4-inch balls.

Soon after the development of double-step liners in the primary mill an attempt was made to adapt the design to the secondary mills. At the time, the importance of proportioning valleys and lifters according to ball size was not fully appreci-

Table 3.—*Manganese-steel Liners in Secondary-mill Shells*

Type	Number of Sets	Average Weight, Lb.	Average Number of Operating Days	Average Weight, per Day, Lb.	Percentage of Original Cost
Block, solid.	2	10,198	272	37.5	100
Shiplap, solid	1	8,694	232	37.5	100
Shiplap, slotted.......	3	8,366	241	34.7	93
Shiplap, spiral, solid..	1	8,700 (est.)	252	34.5	92
Double-step, slotted, narrow lifter......	1	7,515	221	34.0	91
Double-step, slotted, proportioned....	1	9,632	363	26.5	71
	1	9,632	383[a]	25.1	67

[a] Estimated after 293 days of service.

ated, and the initial design was faulty in the following respects: (1) For a ball charge of 2½-in. balls, the valleys were excessively deep and wide—a valley would seat 2½-in. balls completely and accommodate two of them in line of travel; (2) the lifters were only 1¼ in. wide at the top—so narrow that they soon knife-edged and peened excessively; (3) the liners were slotted, which also detracted from the length of life. These liners showed no appreciable reduction in liner cost nor improvement in grinding capacity.

The double-step liners for the secondary mills were redesigned with lifters and valleys proportioned for 2½-in. balls, in the manner that the primary-mill liners were proportioned to 4-in. balls. Slots were still included in the design. These liners lasted a year or longer as compared with about eight months for the shiplap liners, and effected a saving in liner cost of over 30 per cent. It is anticipated that, by omitting the slots, further reduction of costs will follow tests with solid double-step liners.

Grinding Capacity

It has been shown that in the primary mill the block-type liners did not grind well in the early stages, that the undulating type of liners lost grinding capacity in the later stages, and that the double-step liners enabled full grinding capacity from start to finish.

In the secondary mills, when the revised double-step liners were installed, a sustained and substantial increase in grinding capacity was noted. Because the amount of feed to this circuit was limited by the primary mill, advantage was taken of the increased grinding capacity by reducing the ball charge in the secondary mills. Thereby the ball consumption was reduced about 6 per cent and an appreciable but unmeasured saving in power was noted.

In a primary mill with double-step liners at another property, reported results* showed an increase of 12 per cent in grinding capacity as compared with wave-type liners. This enabled an increase of 12 per cent in the production of the plant.

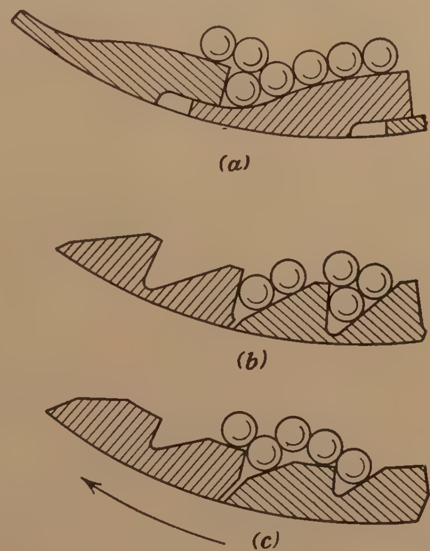

Fig. 4.—Profiles of various secondary-mill-shell liners with arrangement of balls of maximum size.

a, shiplap; *b*, double step with wide valley and narrow lifter; *c*, double step proportioned.

Theory of Double-step Liners

The primary function of the double-step liners designed according to the size of balls used in the charge was to prevent slippage by frequently using grinding balls as keys and the valleys between lifters as keyseats in locking ball charge to mill shell. By spacing the lifters and valleys so that alternate balls in line of travel would lie in valleys and project sufficiently to act as blocks against slippage of adjacent balls resting on lifters (Fig. 4), durable lifters with maximum resistance to slippage were obtained.

The valleys were slightly wider than one ball diameter and were V-shaped, with the trailing wall at an angle of about 55° with the radial leading wall. This permitted

* *Amsco Bulletin* (September 1941).

balls to seat securely and still allowed them to fall freely at the height of the trajectory. Comparatively narrow valleys would not seat the larger balls securely, and steep walls would wedge balls in place. When this happens, the projecting portions of the wedged balls grind off flush with the adjacent lifters, leaving a substantially smooth area on which slippage may take place.

A valley wide enough to accommodate two or more balls would leave room for excessive slippage. This was a contributing cause of the excessive and uneven wear in the valleys of the block-type and shiplap liners.

A sloping leading wall on the lifter was found necessary to secure uniform ball action and full grinding capacity with new liners. A steep leading wall on the lifter, as in the block-type liners, impaired ball action and grinding capacity.

The proportionate width and height of lifters developed into important factors. There was a general tendency in all the liners to show nearly twice as much wear on the leading wall of the lifter as on the top. When the width of lifters in the double-step liners was made one ball diameter and about half as high, they sustained very well. Deviation to narrower lifters caused the negative results with undulating liners in the primary mill and narrow-lifter double-step liners in a secondary mill. Deviation to excessively wide lifters led to the more rapid wear observed along the line of gussets that joined pairs of lifters in the first set of double-step liners in the primary mill.

In addition to the prevention of slippage, the vigorous ball action delivered by the double-step liners is believed to be responsible for the increase in grinding capacity. The positive keying effect between balls and lifters placed little reliance upon any certain kind or quantity of feed to regulate slippage and enhance ball action. The multiplicity of lifters delivered a steady stream of balls and was particularly effective under heavy load; also, fallen balls were quickly picked up and put back to work.

These factors add up to the optimum design of liners as found at Mammoth. In terms of maximum size of balls, the lifters were about one ball-diameter wide at the top. The valleys were in V-section, slightly wider than one ball of maximum size, with radial leading wall and inclined trailing wall forming an angle of about 55°.

Summary and Conclusions

An investigation of ball-mill liners at the Mammoth mill disclosed that mill performance and liner life are substantially dependent upon liner design.

Slippage was found to be a major cause of rapid liner wear and loss of grinding capacity.

A design of double-step liner was developed, based primarily upon the size of balls used in the charge,* which promoted ball action and reduced slippage by using the balls to help key the charge to the mill shell.

The double-step liners reduced total consumption of liners by 45 per cent—a saving at the rate of 28 tons of manganese steel per year in three ball mills. Appreciable savings in consumption of balls and power were accomplished through improved grinding efficiency and capacity. Fewer interruptions for relining and higher over-all grinding capacity permitted increased plant production.

Acknowledgment

The author extends his thanks to the Management of Mammoth-St. Anthony, Ltd., for permission to publish the data herein contained on the Mammoth operation.

* U. S. Patent No. 2274331.

Some Recent Applications of Heavy-media Separation (Sink-float) Processes

By S. J. Swainson,* Member A.I.M.E., S. A. Falconer* and G. B. Walker*

(New York Meeting, February 1943)

During the past few years much interest and attention has been focused on a relatively new method of ore concentration, which utilizes the principles of sink and float and employs as the heavy medium a suspension in water of a finely ground solid, such as galena or ferrosilicon. Although a relatively recent development, much progress has already been made in applying heavy-media separation processes to a considerable number of ores. Eight plants in this country and abroad are now treating approximately 10 million tons of ore annually by heavy-media separation (sink-float) processes, including lead-zinc, lead, iron, garnet, tin and tungsten.

At the annual meeting of the A.I.M.E. in 1941, three interesting papers (unpublished) dealing with heavy-media separation (sink-float) processes of ore concentration as practiced in four commercial units in the United States were presented by Victor Rakowsky, Grover Holt, Elmer Isern and Robert Ammon. These papers gave an excellent exposition of the principles of sink and float, as well as the operating practice in two iron-ore-beneficiation plants, the Harrison and the Merritt, of Butler Brothers, in Minnesota; the Central mill of the Eagle Picher Mining and Smelting Co.; and the Mascot mill, of the American Zinc, Lead and Smelting Co. Augmented by pictorial descriptions and a working model demonstrating the processes, these papers evoked a great deal of interest and discussion. In fact, in reporting the high lights of the meeting in the April 1941 issue of Mining and Metallurgy, Professor Taggart made the statement that "preconcentration by heavy suspension was Miss A.I.M.E. 1941 so far as milling is concerned."

Although to date the greatest application of heavy-media separation from the tonnage point of view has been for the treatment of lead, zinc and iron ores, these processes have a much wider field of usefulness. In this regard, it is to be noted that recently new heavy-media separation plants for treating cassiterite ore and garnet ore have been placed in successful operation. Moreover, numerous other types of ores have been found to respond well to these processes when tested on a semi-commercial scale, and it is expected that in the near future a considerable number of new plants will be erected.

In view of the fact that heavy-media separation processes offer such attractive possibilities, particularly now, when the mining industry is being called on for an even larger production than ever before has been attempted, despite shortages of labor and supplies, it was felt that a description of some of the successful separations that have been worked out during the past year in the Ore Dressing Laboratory of the American Cyanamid Co. on a variety of ores of strategic importance would be of interest. The purpose of this paper, therefore, is to present the essential data relating to investigations of heavy-

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* Ore Dressing Laboratory, American Cyanamid Co., New York, N. Y.

media separation on various types of ores received at the Stamford Laboratory during the past year. For the benefit of those who may not have followed as closely as others the development of heavy-media separation, a general bibliography is given at the end of this paper. The publications listed contain information dealing with the fundamental principles of the processes, appara'us and types of media employed, and with the general fields of usefulness of heavy-media separation (sink-float) processes.

The results of some typical tests on ores containing such valuable minerals as chalcopyrite, malachite, bornite, scheelite, fluorite, magnesite, cassiterite, topaz, wolframite, manganese oxides, siderite, sphalerite and galena are reported in the following pages. It will be apparent that heavy-media separation (sink-float) processes may be applied in the following ways for the beneficiation of a variety of ores in which the valuable mineral constituents have an appreciable difference in specific gravity from the worthless gangue minerals:

1. The production of a finished concentrate and a rejectable waste product in one operation.

2. The production of a finished concentrate and a low-grade reject, which may require additional treatment.

3. The rejection of a finished waste product, leaving a low-grade product for subsequent concentration by other methods.

In general, three types of ore are amenable to heavy-media separation processes; namely:

1. Ores whose valuable constituents are liberated from unwanted mineral at a grind of 10 mesh or coarser.

2. Ores containing valuable minerals in the form of lenses or veins, separated from each other by barren gangue.

3. Ores containing barren gangue that can be rejected after coarse crushing.

From the foregoing it will be apparent that heavy-media separation processes offer a cheap and effective means for making ore out of waste dumps and tailings, by concentrating the valuable constituents into a smaller product of sufficiently high grade to warrant treatment by more expensive methods, such as flotation or cyanidation.

The range of sizes in the feed to these processes can be much greater than the range treated by ordinary methods of concentration such as jigging. The upper size limit is dependent only upon the crushing characteristics of the ore as regards the liberation of the valuable mineral. At one plant, 2-in. ore is being treated economically. A recent development in heavy-media separation processes has made possible the treatment of some types of ore ground to sizes as fine as 48 mesh. Thus, heavy-media separation processes offer the possibility of treating suitable ores from their coarsest liberation side down to the size range usually considered to be suitable for flotation.

For the benefit of those interested, a description of the apparatus and technique used in conducting semicommercial heavy-media separation tests in the Ore Dressing Laboratory of the American Cyanamid Co. is presented in the Appendix (p. 421).

In reviewing the following descriptions of the application of heavy-media separation to various types of ores, it should be borne in mind that the results reported were obtained under conditions simulating actual commercial-scale operations. The type of apparatus, the heavy separating medium, and the technique employed, are the same as those used in commercially operating plants. In most instances the tests were conducted on a continuous scale; that is, the ore was fed continuously, at a previously determined rate, to the separating cone for a sufficient period to obtain reliable data. Under such conditions, the data obtained can be directly translated into full-scale operation, and plants can

safely be designed on the basis of such information.

When designed from experience gained on large-scale operations and after adequate test work in a well-equipped ore-dressing laboratory, heavy-media separation plants can be built quickly and inexpensively, since they are almost entirely assembled from standard units of milling equipment. Operating costs of heavy-media separation plants are lower than those of any other concentrating process. Because a wide variety of heavy media—both ferrous and nonferrous—can be used, heavy-media separation has great scope and flexibility of application.

In this connection, a new pilot plant for heavy-media separation, utilizing commercial equipment, has just been put into operation by the American Cyanamid Co. at Stamford, Conn. This new plant serves as a development and proving ground for equipment used in heavy-media separation processes, as well as for large-scale testing of carload lots of ore where sampling or other conditions require such tests.

EXAMPLE 1.—*Copper-tungsten Ore*

Ore: Copper-tungsten ore from Washington
Mineralization: Chalcopyrite, malachite, bornite, arsenopyrite, pyrite, and scheelite with quartz and calcite and gold
Analysis: Cu, 1.68 per cent; WO_3, 1.12; Au, 0.005 oz. per ton; Ag, 0.92

Distribution of Values after Crushing Ore to Minus 1 Inch

Product	Wt., Per Cent	Assays				Distribution, Per Cent			
		Cu, Per Cent	WO_3, Per Cent	Au, Oz. per Ton	Ag, Oz. per Ton	Cu	WO_3	Au	Ag
−1-in + 10-mesh	70.29	1.81	0.35	0.003	0.85	75.48	21.68	44.22	65.36
−10-mesh	29.71	1.39	2.97	0.010	1.07	24.52	78.32	55.78	34.64
Calculated head	100.00	1.68	1.12	0.005	0.92	100.00	100.00	100.00	100.00

Apparatus Used in Heavy-media Separation: 20-in. continuous cone
Medium Used: Ferrosilicon, minus 65 mesh; 50 per cent minus 325 mesh
Size Fraction Ore Treated: Minus 1-in. plus 10-mesh

Results of Separation at 2.80 Specific Gravity on Minus 1-inch Plus 10-mesh Ore

Product	Weight, Per Cent		Assays					Distribution in Size, Per Cent					Distribution, Per Cent, in Total Ore				
	Of Size	Of Total	Total Cu, Per Cent	Oxide Cu, Per Cent	WO_3, Per Cent	Au, Oz. per Ton	Ag, Oz. per Ton	Total Cu	Oxide Cu	WO_3	Au	Ag	Total Cu	Oxide Cu	WO_3	Au	Ag
Total ore		100.00	1.70	0.25	1.12	0.005	0.95						100.0	100.0	100.0	100.0	100.0
−1 in. +10-mesh cone feed	100.00	70.29	1.83	0.17	0.34	0.003	0.90	100.00	100.0	100.00	100.0	100.0	75.65	47.40	21.24	43.64	66.60
Sink in 2.80	10.91	7.67	14.77	0.51	2.94	0.030	6.96	88.29	32.44	94.74	100.00	84.20	66.79	15.38	20.13	43.64	56.08
Float on 2.80	89.09	62.62	0.24	0.13	0.02	Nil	0.16	11.71	67.56	5.26	Nil	15.80	8.86	32.02	1.11	Nil	10.52
−10-mesh (not treated)		29.71	1.39	0.45	2.97	0.010	1.07						24.35	52.60	78.76	56.36	33.40

The results of Example 1 show that of the total minus 1-in. plus 10-mesh ore fed to the cone, 10.91 per cent was recovered as a sink product in 2.80 sp. gr. This product assayed 14.77 per cent copper and 2.94 per cent tungstic oxide, and contained 88.29 per cent of the copper and 94.74 per cent of the tungsten present in the cone feed. The float product was low enough in grade to be discarded, and it represented almost two thirds, by weight, of the original ore before screening.

EXAMPLE 2.—*Fluorite Ore*

Ore: Fluorspar ore from Rosiclare, Illinois, district
Mineralization: Fluorite in gangue of limestone, calcite and quartz, with minor amounts of galena, sphalerite and chalcopyrite
Analysis: CaF_2, 56.44 per cent; $CaCO_3$, 23.60; SiO_2, 10.80; Pb, 0.27; Zn, 0.68

Distribution after Crushing to Minus 1 Inch

Product	Weight, Per Cent	Assays, Per Cent			Distribution, Per Cent		
		CaF_2	$CaCO_3$	SiO_2	CaF_2	$CaCO_3$	SiO_2
Feed	100.0	56.44	23.60	10.80	100.0	100.0	100.0
−1-in. + 10-mesh	67.33	51.39	26.44	13.09	61.20	75.44	81.58
−10-mesh	32.67	67.15	17.74	6.09	38.80	24.56	18.42

Apparatus Used in Heavy-media Separation: 20-in. continuous cone
Medium Used: Ferrosilicon, minus 65 mesh; 53 per cent minus 325 mesh
Size Fraction Ore Treated: Minus 1-in. plus 10-mesh

Results of Heavy-media Separation

Product	Weight, Per Cent		Assays, Per Cent			Distribution, Per Cent					
						Of Size			Of Total Ore		
	Of Size	Of Total	CaF_2	$CaCO_3$	SiO_2	CaF_2	$CaCO_3$	SiO_2	CaF_2	$CaCO_3$	SiO_2
Results of Separation at 2.88 Specific Gravity											
Original ore		100.00	61.25	24.40	10.74				100.00	100.00	100.00
−1-in. +10-mesh cone feed	100.00	64.47	54.78	27.64	13.35	100.0	100.00	100.00	57.65	73.03	80.12
Sink in 2.88	54.43	35.09	90.78	0.98	4.39	90.21	1.93	17.90	52.01	1.41	14.34
Float on 2.88	45.57	29.38	11.77	59.48	24.05	9.79	98.07	82.10	5.64	71.62	65.78
−10-mesh (not treated)		35.53	73.00	18.52	6.01				42.35	26.97	19.88
Results of Separation at 2.95 Specific Gravity											
Original ore		100.00	59.94	21.62	10.83				100.00	100.00	100.00
−1-in. +10-mesh cone feed	100.00	67.33	56.42	23.51	13.13	100.00	100.00	100.00	63.40	73.20	81.62
Sink in 2.95	53.19	35.81	92.16	0.81	4.94	86.88	1.84	20.01	55.06	1.35	16.33
Float on 2.95	46.81	31.52	15.81	49.29	22.43	13.12	98.16	79.99	8.34	71.85	65.29
−10-mesh (not treated)		32.67	67.15	17.74	6.09				36.60	26.80	18.38
Results of Separation at 3.00 Specific Gravity											
Original ore		100.00	58.69	24.32	11.13				100.00	100.00	100.00
−1-in. +10-mesh cone feed	100.00	67.33	54.59	27.52	13.57	100.00	100.00	100.00	62.62	76.17	82.12
Sink in 3.00	45.20	30.43	92.43	0.77	3.72	76.53	1.26	12.40	47.92	0.96	10.18
Float on 3.00	54.80	36.90	23.38	49.58	21.69	23.47	98.74	87.60	14.70	75.21	71.94
−10-mesh (not treated)		32.67	67.15	17.74	6.09				37.38	23.83	17.88

From the figures shown in Example 2, it is apparent that heavy-media separation offers very good possibilities on this fluorite ore. By treating the minus 1-in. plus 10-mesh fraction of the ore at 2.95 to 3.00 sp. gr., a sink product was obtained of sufficiently high grade to be marketed without further treatment. About 50 per cent of the total fluorite in the original ore was recovered in this sink product. The float product and the minus 10-mesh product, being too high grade for discard, would normally be combined, ground, and subjected to froth flotation.

EXAMPLE 3.—*Fluorspar Ore*

Ore: Plus 10-mesh ore from Rosiclare, Illinois, district
Mineralization: Fluorite in gangue of limestone, calcite and quartz, with minor amounts of galena, sphalerite and chalcopyrite
Analysis: CaF_2, 60.32 per cent; $CaCO_3$, 22.28; SiO_2, 15.20; Pb, 0.14; Zn, 0.20
Distribution: Not determined, −¾-in. +10-mesh as received

Results of Separation at 2.88 Specific Gravity

Product	Weight, Per Cent	Assays, Per Cent					Distribution, Per Cent				
		CaF_2	$CaCO_3$	SiO_2	Pb	Zn	CaF_2	$CaCO_3$	SiO_2	Pb	Zn
Cone feed (all of ore as received)	100.0	60.32	22.28	15.20	0.14	0.20	100.0	100.0	100.0	100.0	100.0
Sink in 2.88	59.41	92.18	1.04	5.29	0.22	0.22	90.79	2.77	20.68	97.04	65.15
Float on 2.88	40.51	13.68	53.36	29.71	0.10	0.17	9.21	97.23	79.32	2.96	34.85

Apparatus Used in Heavy-media Separation: 20-in. continuous cone
Medium Used: Ferrosilicon, minus 65 mesh; 50 per cent minus 325 mesh
Size Fraction Ore Treated: Minus 1-in. plus 10-mesh

The results shown in Example 3 indicate the possibility of utilizing heavy-media separation to produce a high-grade fluorspar concentrate while at the same time rejecting 97 per cent of the calcite and almost 80 per cent of the silica in this sized ore sample. In view of the high recovery of fluorite shown, it is doubtful whether further treatment of the float product would be economical.

EXAMPLE 4.—*Fluorspar Tailing*

Ore: Jig tailing from Rosiclare, Illinois, district
Mineralization: Jig tailing containing limestone, calcite, quartz and fluorite. Fluorite largely free but some associated with quartz
Analysis: CaF_2, 22.11 per cent; $CaCO_3$, 32.76; SiO_2, 41.85

Distribution of CaF_2, $CaCO_3$, and SiO_2 on Sample as Received[a]

Product	Weight, Per Cent	Assays, Per Cent			Distribution, Per Cent		
		CaF_2	$CaCO_3$	SiO_2	CaF_2	$CaCO_3$	SiO_2
Original feed	100.00	22.11	32.76	41.85	100.00	100.00	100.00
¼-in.	40.54	15.35	32.65	46.30	28.13	40.41	44.83
−¼-in. +10-mesh	45.97	25.21	33.63	39.42	52.42	47.16	43.27
−10-mesh	13.49	30.87	29.21	35.70	19.45	12.43	11.90

Apparatus Used in Heavy-media Separation: 20-in. semicontinuous cone
Medium Used: Ferrosilicon, minus 65 mesh; 61 per cent minus 325 mesh
Size Fraction Material Treated: Tests run on plus ¼-in. fraction and minus ¼-in. plus 10-mesh fraction separately, also on total plus 10-mesh material as received

[a] Sample consisted of minus ⅜-in. particles to fines.

Results of Heavy-media Separation

Product	Weight, Per Cent		Assays, Per Cent			Distribution, Per Cent					
						Of Size			Of Total Ore		
	Of Size	Of Total	CaF_2	$CaCO_3$	SiO_2	CaF_2	$CaCO_3$	SiO_2	CaF_2	$CaCO_3$	SiO_2
Results of Separation on Plus ¼-inch Feed at 2.90 Specific Gravity											
Original feed		100.00	21.79	32.92	41.63				100.00	100.00	100.00
Cone feed	100.00	40.54	15.66	33.25	47.71	100.00	100.00	100.00	29.13	40.95	46.47
Sink in 2.90	6.84	2.77	90.22	1.50	8.27	39.40	0.31	1.19	11.47	0.13	0.55
Float on 2.90	93.16	37.77	10.19	35.58	50.60	60.60	99.69	98.81	17.66	40.82	45.92
Results of Separation on Minus ¼-inch Plus 10-mesh Feed at 2.90 Specific Gravity											
Original feed		100.00	21.79	32.92	41.63				100.00	100.00	100.00
Cone feed	100.00	45.97	24.54	33.71	38.00	100.00	100.00	100.00	51.76	47.08	41.96
Sink in 2.90	19.32	8.88	91.68	1.61	6.71	72.19	0.92	3.41	37.36	0.43	1.43
Float on 2.90	80.68	37.09	8.46	41.40	45.49	27.81	99.08	96.59	14.40	46.65	40.53
Combined Results of Separation on Plus ¼-inch and Minus ¼-inch Plus 10-mesh Feeds at Specific Gravity 2.90											
Original feed		100.00	21.79	32.92	41.63				100.00	100.00	100.00
Cone feeds	100.00	86.51	20.38	33.50	42.55	100.00	100.00	100.00	80.89	88.03	88.43
Combined sinks	13.47	11.65	91.33	1.59	7.08	60.37	0.64	2.24	48.83	0.56	1.98
Combined floats	87.53	74.86	9.33	38.47	47.81	39.63	99.36	97.76	32.06	87.47	86.45
−10-mesh (not treated)		13.49	30.87	29.21	35.70				14.11	11.97	11.57
Results of Separation on Total Plus 10-mesh Feed at 2.90 Specific Gravity											
Original feed		100.00	22.36	30.89	42.25				100.00	100.00	100.00
Cone feed	100.00	86.51	21.04	31.15	43.50	100.00	100.00	100.00	81.38	87.75	88.66
Sink in 2.90	14.27	12.34	88.55	1.07	7.77	60.06	0.49	2.55	48.87	0.43	2.26
Float on 2.90	85.73	74.17	9.80	36.16	49.45	39.94	99.51	97.45	32.51	86.82	86.40
−10-mesh (not treated)		13.49	30.87	29.21	35.70				18.62	12.25	11.34

Example 4 shows that better results were obtained on the minus ¼-in. plus 10-mesh fraction than on the minus ½-in. plus ¼-in. fraction. This is due to the better liberation of the fluorite in the finer sized material.

It is also interesting to note that, in this particular example, a somewhat higher content of fluorite in the sink product was obtained by separate treatment of the minus ½-in. plus ¼-in. and the minus ¼-in. plus 10-mesh fractions than with treatment of the minus ½-in. plus 10-mesh portion.

It is evident from these results that heavy-media separation at 2.90 sp. gr. did a very creditable job on this low-grade jig-tailing sample. Approximately one half of the fluorite in the original material was recovered as a sink product assaying about 90 per cent CaF_2, and therefore of suitable grade for marketing as flux.

EXAMPLE 5.—*Magnesite Ore*

Ore: Magnesite ore from Nevada
Mineralization: Magnesite, dolomite and dike rock
Analysis: CaO, 8.41 per cent; insoluble, 3.06

Distribution of CaO and Insoluble after Crushing to Minus 1 Inch

Product	Weight, Per Cent	Assays, Per Cent		Distribution, Per Cent	
		CaO	Insoluble	CaO	Insoluble
Total feed	100.00	8.41	3.06	100.00	100.00
−1-in. +10-mesh	72.51	8.06	1.55	69.53	36.79
−10-mesh	27.49	9.32	7.04	30.47	63.21

Apparatus Used in Heavy-media Separation: 20-in. continuous cone
Medium Used: Ferrosilicon, minus 65 mesh; 53 per cent minus 325 mesh
Size Fraction Ore Treated: Minus 1-in. plus 10-mesh

Results of Heavy-media Separation

Product	Weight, Per Cent		Assays, Per Cent		Distribution, Per Cent			
					In Size		In Total	
	Of Size	Of Total	CaO	Insoluble	CaO	Insoluble	CaO	Insoluble
Results of Separation at 2.92 Specific Gravity								
Total ore		100.00	8.41	3.06			100.00	100.00
Cone feed	100.00	72.51	8.06	1.55	100.00	100.00	69.53	36.79
Sink in 2.92	50.37	36.52	3.39	0.73	21.20	23.86	14.75	8.79
Float on 2.92	49.63	35.99	12.80	2.38	78.80	76.14	54.78	28.00
−10-mesh (not treated)		27.49	9.32	7.04			30.47	63.21
Results of Separation at 2.94 Specific Gravity								
Total ore		100.00	8.41	3.06			100.00	100.00
Cone feed	100.00	72.51	8.06	1.55	100.00	100.00	69.53	36.79
Sink in 2.94	28.83	20.90	2.97	0.69	10.62	12.77	7.39	4.71
Float on 2.94	71.17	51.61	10.13	1.90	89.38	87.23	61.14	32.08
−10-mesh (not treated)		27.49	9.32	7.04			30.47	63.21
Results of Separation at 2.97 Specific Gravity								
Total ore		100.00	8.41	3.06			100.00	100.00
Cone feed	100.00	72.51	8.06	1.55	100.00	100.00	69.53	36.79
Sink in 2.97	7.09	5.14	2.94	0.78	2.58	3.55	1.80	1.31
Float on 2.97	92.91	67.37	8.45	1.61	97.42	96.45	66.73	35.46
−10-mesh (not treated)		27.49	9.32	7.04			30.47	63.21

The results shown in Example 5 indicate the possibility of utilizing heavy-media separation to reject almost 90 per cent of the lime and insoluble material in the minus 1-in. plus 10-mesh fraction of this relatively low-grade magnesite ore. The sink product resulting from the separation at 2.94 sp. gr. represented almost 29 per cent of the weight of the cone feed. This sink product would be sufficiently high in grade for some uses. The float product and the minus 10-mesh fraction could be recombined, ground, and subjected to froth-flotation treatment.

EXAMPLE 6.—*Magnesite Ore*

Ore: Magnesite ore from Nevada
Mineralization: Magnesite with dolomite and rhyolite dike material. Fine to coarse dissemination
Analysis: CaO, 9.55 per cent; insoluble, 11.63

Distribution of CaO and Insoluble after Crushing Ore to Minus 1 Inch

Product	Weight, Per Cent	CaO, Per Cent	Insoluble, Per Cent
Original ore	100.00	9.55	11.63
−1 + ¼ in.	52.20	9.52	11.13
−¼ + 10-mesh	21.44	9.72	13.73
−10-mesh	26.36	9.45	10.90

Apparatus Used in Heavy-media Separation: Semicontinuous 20-in. cone
Medium Used: Ferrosilicon
Size Fraction Ore Treated: Minus 1-in. plus 10-mesh

Results of Separation on Minus 1-inch Plus ¼-inch and ¼-inch Plus 10-mesh Ore at 2.95 Specific Gravity

Product	Weight, Per Cent		Assays, Per Cent		Distribution, Per Cent			
					In Size		In Total	
	Of Size	Of Total	CaO	Insoluble	CaO	Insoluble	CaO	Insoluble
Original ore		100.00	9.94	11.34			100.00	100.00
Cone feed −1 + ¼ in.	100.00	52.20	9.95	10.71	100.00	100.00		
Sink in 2.95	40.99	21.40	2.18	1.10	8.94	4.20	4.73	2.11
Float on 2.95	59.01	30.80	15.36	17.38	91.06	95.80	47.59	47.18
Cone feed −¼-in. + 10-mesh	100.00	21.44	10.52	13.42	100.00	100.00		
Sink in 2.95	28.23	6.05	4.02	1.77	10.74	3.73	2.41	0.97
Float on 2.95	71.77	15.39	13.08	18.00	89.26	96.27	20.22	24.43
−10-mesh (not treated)		26.36	9.45	10.90			25.05	25.31
Recapitulation. Separation on Minus 1-inch Plus 10-mesh Ore								
Original ore		100.00	9.94	11.34			100.00	100.00
Combined cone feed	100.00	73.64	10.12	11.50	100.00	100.00	74.95	74.69
Combined sink in 2.95	37.28	27.45	2.59	1.28	9.53	4.12	7.14	3.08
Combined float on 2.95	62.72	46.19	14.59	17.58	90.47	95.88	67.81	71.61
−10-mesh (not treated)		26.36	9.45	10.90			25.05	25.31

Example 6 shows that approximately the same percentage rejections of both lime and insoluble material were obtained in the finer sizes as on the coarser sizes. The recapitulated figures show that almost ⅔ of the minus 1-in. plus 10-mesh fraction of this magnesite ore was rejected in a float product containing slightly more than 90 per cent of the lime and about 96 per cent of the insoluble material in the feed to the cone. The combined sink product assayed 2.59 per cent CaO and 1.28 per cent insoluble, these figures representing a very substantial reduction from the figures of 9.94 per cent CaO and 11.34 per cent insoluble in the cone feed.

Example 7.—*Magnesite Ores*

Ore: Magnesite ores from Washington
Mineralization: Magnesite intermixed with dolomite, calcite, talc, serpentine, quartz and shale
Analyses of Samples:

Sample No.	CaO, Per Cent	Insoluble, Per Cent
A-1	5.42	7.64
A-2	5.06	6.83
A-3	7.23	3.82
F-1	2.41	8.91
F-23	3.42	5.34
F-4	3.09	10.00
F-5	7.30	4.78
F-6	3.03	4.49

Distribution of CaO and Insoluble after Crushing Sample to Minus 1 Inch

Sample No.	Weight, Per Cent			Assays, Per Cent						Distribution, Per Cent					
	Orig.	−1 + ¼-in.	−¼-in.	Total Ore		−1 + ¼-in.		−¼-in.		Total Ore		−1 + ¼-in.		−¼-in.	
				CaO	Insol.	CaO	Insol.	CaO	Insol.	CaO	Insol.	CaO	Insol.	CaO	Insol.
A-1	100	62.89	37.11	5.42	7.64	5.94	7.43	4.62	8.00	100.00	100.00	68.29	61.09	31.71	38.91
A-2	100	70.15	29.85	5.06	6.83	5.28	6.55	4.52	7.48	100.00	100.00	73.29	67.30	26.71	32.70
A-3	100	65.84	34.16	7.23	3.82	8.16	3.83	5.64	3.80	100.00	100.00	73.47	66.03	26.53	33.97
F-1	100	75.10	24.90	2.41	8.91	2.02	8.07	3.58	11.40	100.00	100.00	62.83	68.10	37.17	31.90
F-23	100	66.79	33.21	3.42	5.34	3.51	5.19	3.22	5.64	100.00	100.00	68.68	64.92	31.32	35.08
F-4	100	69.67	30.33	3.09	10.00	3.62	10.17	1.88	9.60	100.00	100.00	81.43	70.86	18.57	29.14
F-5	100	72.15	27.85	7.30	4.78	7.80	4.49	6.00	5.52	100.00	100.00	77.09	67.83	22.91	32.17
F-6	100	58.02	41.98	3.03	4.49	3.12	3.91	2.90	5.28	100.00	100.00	59.81	50.55	40.19	49.45

Object of Separation: (1) to produce "high-grade" rock containing less than 1.5 per cent CaO and 2.0 per cent SiO_2; (2) to produce "Standard" rock containing less than 2.5 per cent CaO and 3.2 per cent SiO_2.

Results of Separation on Various Ores

Sample No.	Product	Weight, Per Cent		Assays, Per Cent			Distribution, Per Cent				
							In Size			In Total	
		In Size	In Total	CaO	Insol.	SiO_2	CaO	Insol.	SiO_2	CaO	Insol.
A-1	Original ore		100.00	5.85	7.27					100.00	100.00
	$-1 + \frac{1}{4}$-in. cone feed	100.00	62.89	6.58	6.84	5.64	100.00	100.00	100.00	70.72	59.16
	Sink in 2.90	58.04	36.50	2.94	4.70	3.64	25.92	39.86	36.46	18.32	23.58
	Float on 2.90	41.96	26.39	11.62	9.80	8.40	74.08	60.14	63.54	52.40	35.58
	$-\frac{1}{4}$-in. (not treated)		37.11	4.62	8.00					29.28	40.84
A-2	Original ore		100.00	4.99	6.91					100.00	100.00
	$-1 + \frac{1}{4}$-in. cone feed	100.00	70.15	5.19	6.67		100.00	100.00		72.96	67.70
	Sink in 2.95	53.33	37.41	1.64	3.30	2.00	16.87	26.39		12.31	17.87
	Float on 2.95	46.67	32.74	9.24	10.52		83.13	73.61		60.65	49.83
	$-\frac{1}{4}$-in. (not treated)		29.85	4.52	7.48					27.04	32.30
A-3	Original ore		100.00	5.88	3.29					100.00	100.00
	$-1 + \frac{1}{4}$-in. cone feed	100.00	65.84	6.01	3.06		100.00	100.00		67.27	60.55
	Sink in 2.95	42.77	28.16	1.59	2.06	1.64	11.31	28.83		7.61	17.63
	Float on 2.95	57.23	37.68	9.32	3.80		88.69	71.17		59.66	42.92
	-1-in. (not treated)		34.16	5.64	3.80					32.73	39.45
F-1	Original ore		100.00	2.68	7.48					100.00	100.00
	$-1 + \frac{1}{4}$-in. cone feed	100.00	75.10	2.38	6.17		100.00	100.00		66.69	62.02
	Sink in 2.91	83.14	62.44	1.28	6.35	2.27	44.78	85.52		29.87	53.04
	Float on 2.91	16.86	12.66	7.78	5.30		55.22	14.48		36.82	8.98
	$-\frac{1}{4}$-in. (not treated)		24.90	3.56	11.40					33.31	37.98
F-23	Original ore		100.00	3.07	5.21					100.00	100.00
	$-1 + \frac{1}{4}$-in. cone feed	100.00	66.79	3.00	5.00		100.00	100.00		65.18	64.07
	Sink in 2.92	71.98	48.08	1.95	4.82	2.84	46.86	69.39		30.55	44.46
	Float on 2.92	28.02	18.71	5.68	5.46		53.14	30.61		34.63	19.61
	$-\frac{1}{4}$-in. (not treated)		33.21	3.22	5.64					34.82	35.93
F-4	Original ore		100.00	2.75	9.71					100.00	100.00
	$-1 + \frac{1}{4}$-in. cone feed	100.00	69.67	3.13	9.75		100.00	100.00		79.26	70.00
	Sink in 2.93	65.80	45.84	2.11	7.60	2.14	44.42	51.26		35.19	35.89
	Float on 2.93	34.20	23.83	5.08	13.90		55.58	48.74		44.07	34.11
	$-\frac{1}{4}$-in. (not treated)		30.33	1.88	9.60					20.74	30.00
F-5	Original ore		100.00	7.16	4.29					100.00	100.00
	$-1 + \frac{1}{4}$-in. cone feed	100.00	72.15	7.60	3.82		100.00	100.00		76.65	64.20
	Sink in 2.95	62.70	45.24	1.99	3.26	1.84	16.41	53.52		12.58	34.36
	Float on 2.95	37.30	26.91	17.04	4.76		83.59	46.48		64.07	29.84
	$-\frac{1}{4}$-in. (not treated)		27.85	6.00	5.52					23.35	35.80
F-6	Original ore		100.00	3.23	4.38					100.00	100.00
	$-1 + \frac{1}{4}$-in. cone feed	100.00	58.02	3.48	3.73		100.00	100.00		62.39	49.37
	Sink in 2.95	69.68	40.43	1.12	3.18	2.02	22.43	59.47		14.00	29.37
	Float on 2.95	30.32	17.59	8.90	4.98		77.57	40.53		48.39	20.00
	$-\frac{1}{4}$-in. (not treated)		41.98	2.90	5.28					37.61	50.63

From the results reported in Example 7, it is apparent that by selection of the optimum specific gravity of medium it was possible to produce from the magnesite ores treated, sink products meeting the specifications set for "high-grade" rock (less than 1.5 per cent CaO and 2.0 per cent SiO_2) in some instances, or, in most cases, at least the specifications set for "Standard Grade" rock (less than 2.5 per cent CaO and 3.27 per cent SiO_2). It is to be noted that the average grade of the sink product of all eight samples is 1.83 per cent CaO and 2.30 per cent SiO_2. Rejection of lime into the float product in these tests ranged from 53 to 88 per cent of the total in the cone feed. Rejection of insoluble into the float product ranged from a low of 14 per cent to a high of 73 per cent.

The results shown in this example again indicate the very attractive possibilities of utilizing heavy-media separation (sink-float) processes for beneficiating off-grade magnesite ores.

EXAMPLE 8.—*Manganese Ore*

Ore: Manganese ore from Arizona
Mineralization: Manganese oxides, calcite, siliceous gangue
Analysis: Mn, 20.5 per cent; Fe, 3.91; CaO, 16.06; insoluble, 22.56

Distribution of Manganese after Crushing to Minus 1 Inch

Product	Weight, Per Cent	Mn, Per Cent	Distribution Mn, Per Cent
Original ore	100.00	20.3	100.00
−1 + ½-in.	29.04	22.25	31.82
−½-in. + 10-mesh	43.39	20.10	42.96
−10-mesh	27.57	18.58	25.22

Apparatus Used in Heavy-media Separation: 20-in. continuous cone
Medium Used: Ferrosilicon, −65 mesh; 50 per cent minus 325 mesh
Size Fraction Ore Treated: Minus 1-in. + 10-mesh

Results of Separation at 2.80 Specific Gravity

Product	Weight, Per Cent		Assays, Per Cent				Distribution, Per Cent							
							In Size				In Total			
	In Size	In Total	Mn	Fe	CaO	Insol.	Mn	Fe	CaO	Insol.	Mn	Fe	CaO	Insol.
Original feed		100.00	20.30	3.51	16.61	21.55					100.00	100.00	100.00	100.00
−1-in. + 10-mesh cone feed	100.00	72.43	20.95	3.37	17.93	18.48	100.00	100.00	100.00	100.00	74.78	69.52	78.09	63.81
Sink in 2.80	51.37	37.21	32.63	3.31	13.52	7.63	80.00	50.44	38.88	20.65	59.80	35.05	30.28	13.18
Float on 2.80	48.63	35.22	8.62	3.43	22.54	30.97	20.00	49.56	61.12	79.35	14.98	34.47	47.81	50.63
−10-mesh (not treated)		27.57	18.58	3.89	13.20	28.28					25.22	30.48	21.91	36.19
Total sink after calcining		31.26	38.85	3.94	16.10	9.08								

The results of Example 8 show that when heavy-media separation at 2.80 sp. gr. was applied to this low-grade manganese ore a sink product was obtained that assayed 32.63 per cent Mn, 3.31 per cent Fe, 13.52 per cent CaO and only 7.63 per cent insoluble. This product after calcining was of marketable grade. Approximately 60 per cent of the total manganese in the ore and 80 per cent of the manganese in the cone feed reported in this sink product. The float product was sufficiently low in manganese to be discardable. About 15 per cent of the total manganese in the ore was in this product. The minus 10-mesh ore, not treated by heavy-media separation, could probably be treated at a profit by tabling and/or flotation.

EXAMPLE 9.—*Manganese Ore*

Ore: Manganese from Eastern Tennessee
Mineralization: Pyrolusite and other manganese oxide minerals with limonite, chert, quartzite and clay. Apparently clean nodules may contain many exceedingly small particles of siliceous gangue
Analysis of ore: Mn, 27.89 per cent; Fe, 10.19; insoluble, 29.42

Distribution of Manganese after Crushing to Minus 1 Inch

Product	Weight, Per Cent	Mn, Per Cent	Distribution Mn, Per Cent
−1-in. + 10-mesh	82.53	29.21	86.39
−10-mesh	17.47	21.67	13.61
Total	100.00	27.89	100.00

Apparatus Used in Heavy-media Separation: 20-in. continuous cone
Medium Used: Ferrosilicon, minus 65 mesh; 50 per cent minus 325 mesh
Size Fraction Ore Treated: Minus 1-in. plus 10-mesh

Results of Heavy-media Separation

Product	Weight, Per Cent		Assays, Per Cent			Distribution, Per Cent					
						In Size			In Total Ore		
	In Size	In Total	Mn	Fe	Insol.	Mn	Fe	Insol.	Mn	Fe	Insol.
Results of Separation at 2.90 Specific Gravity											
Original feed		100.00	29.21	10.16	27.21				100.00	100.00	100.00
−1-in. + 10-mesh cone feed	100.00	82.53	30.81	9.62	25.23	100.00	100.00	100.00	87.04	78.09	76.53
Sink in 2.90	67.25	55.50	40.49	9.56	8.50	88.38	66.86	22.65	76.93	52.21	17.33
Float on 2.90	32.75	27.03	10.93	9.73	59.60	11.62	33.14	77.35	10.11	25.88	59.20
−10-mesh (not treated)		17.47	21.67	12.75	36.54				12.96	21.91	23.47
Specific Gravity: top 2.90, bottom 3.06											
Results of Separation at 3.00 Specific Gravity											
Original feed		100.00	26.57	10.23	31.63				100.00	100.00	100.00
−1-in. + 10-mesh cone feed	100.00	82.53	27.61	9.70	30.59	100.00	100.00	100.00	85.75	78.23	79.82
Sink in 3.00	49.21	40.61	41.94	8.31	8.16	74.75	42.17	13.13	64.09	32.99	10.48
Float on 3.00	50.79	41.92	13.73	11.04	52.32	25.25	57.83	86.87	21.66	45.24	69.34
−10-mesh (not treated)		17.47	21.67	12.75	36.54				14.25	21.77	20.18
Specific Gravity: top 3.00, bottom 3.13											

The data given in Example 9 show that heavy-media separation at 2.90 sp. gr. effected a high recovery of manganese in the minus 1-in. plus 10-mesh cone feed and at the same time the sink product assayed over 40 per cent manganese and only 8.5 per cent insoluble. It is interesting to note that separation at a higher specific gravity raised the grade of the sink about 1.5 per cent manganese but lowered the insoluble only 0.34 per cent. Production of a sink product of metallurgical grade from the coarser sizes of this ore would be virtually impossible because apparently pure particles of manganese oxides contain many exceedingly small particles of quartz.

Example 10.—*Lead-zinc Dump Ore*

Ore: Lead-zinc dump ore from western United States
Mineralization: Sphalerite, argentiferous galena, and pyrite, associated with slate and granitic gangue. In weathering, small amounts of cerussite and anglesite formed on surfaces of galena particles
Analysis: Pb, 1.87 per cent; Zn, 5.34 per cent; Ag, 2.68 oz. per ton

Distribution of Values after Crushing to Minus 1 Inch

Product	Weight, Per Cent	Assays			Distribution, Per Cent		
		Pb, Per Cent	Zn, Per Cent	Ag, Oz. per Ton	Pb	Zn	Ag
Original ore	100.00	1.87	5.34	2.68	100.00	100.00	100.00
−1-in. + 10-mesh	52.84	1.55	5.10	2.89	43.79	50.42	56.90
−10-mesh	47.16	2.23	5.62	2.45	56.21	49.58	43.10

Apparatus Used in Heavy-media Separation: 20-in. continuous cone
Medium Used: Ferrosilicon, minus 65 mesh; 53 per cent minus 325 mesh
Size Fraction Ore Treated: Minus 1-in., plus 10-mesh

Results of Separation at 2.90 Specific Gravity

Product	Weight, Per Cent		Assays			Distribution, Per Cent					
						In Size			In Total		
	In Size	In Total	Pb, Per Cent	Zn, Per Cent	Ag, Oz. per Ton	Pb	Zn	Ag	Pb	Zn	Ag
Original ore		100.00	1.87	5.34	2.68				100.00	100.00	100.00
−1-in. + 10-mesh cone feed	100.00	52.84	1.55	5.10	2.89	100.00	100.00	100.00	43.79	50.42	56.90
Sink in 2.90	32.91	17.39	4.67	13.87	8.18	99.16	89.47	93.26	43.42	45.11	53.06
Float on 2.90	67.09	35.45	0.02	0.80	0.29	0.84	10.53	6.74	0.37	5.31	3.84
−10-mesh (not treated)		47.16	2.23	5.62	2.45				56.21	49.58	43.10
Combined sink and −10-mesh ore		64.55	2.89	7.84	3.99				99.63	94.69	96.16
Reject		35.45	0.02	0.80	0.29				0.37	5.31	3.84

The data given in Example 10 demonstrate clearly how heavy-media separation processes may be utilized as a preconcentration device to treat low-grade dump ores. In this particular instance more than one third of the total weight of the ore was rejected in the float product resulting from separation at 2.90 sp. gr. Over 99 per cent of the lead, 94 per cent of the zinc, and 96 per cent of the silver were recovered in the sink product.

EXAMPLE 11.—*Siderite Iron Ore*

Ore: Iron ore from Canada
Mineralization: Siderite and quartzite
Analysis: Fe, 23.97 per cent; SiO_2, 32.11; Insoluble, 33.36; S, 0.68
Specific Gravity: Siderite, 3.57–3.70; quartz, 2.7–2.8

Distribution of Values after Crushing to Minus 2 Inches

Product	Weight, Per Cent	Assays, Per Cent				Distribution, Per Cent			
		Fe	SiO_2	Insol.	S	Fe	SiO_2	Insol.	S
−2 + 1-in.	72.42	23.94	32.12	33.24	0.62	72.30	72.44	72.18	66.17
−1-in. + 10-mesh	23.11	24.94	30.79	32.34	0.84	23.65	22.17	22.42	27.95
−10-mesh	4.47	21.74	38.60	40.36	0.78	4.05	5.39	5.40	5.88
Total	100.00	23.97	32.11	33.36	0.68	100.00	100.00	100.00	100.00

Apparatus Used in Heavy-media Separation: 20-in. continuous cone
Medium Used: Ferrosilicon, minus 65 mesh; 60 per cent minus 325 mesh
Size Fractions Ore Treated: minus 2 plus 1-in.; minus 1-in. + 10-mesh

Results of Separation at 2.96 Specific Gravity

Product	Weight, Per Cent		Assays, Per Cent				Distribution, Per Cent							
							Of Size				Of Total			
	In Size	In Total	Fe	SiO_2	Insol.	S	Fe	SiO_2	Insol.	S	Fe	SiO_2	Insol.	S
Original ore		100.00	23.97	32.11	33.36	0.68					100.00	100.00	100.00	100.00
−2 + 1-in. cone feed	100.00	72.42	23.94	32.12	33.24	0.62	100.00	100.00	100.00	100.00	72.30	72.44	72.18	66.17
Sink in 2.96	66.21	47.95	33.62	7.90	9.00	0.87	92.98	16.28	17.93	93.55	67.25	11.80	12.95	61.76
Float on 2.96	33.79	24.47	4.96	79.58	80.74	0.11	7.02	83.72	82.07	6.45	5.05	60.64	59.23	4.41
−1-in. + 10-mesh cone feed	100.00	23.11	24.94	30.79	32.34	0.84	100.00	100.00	100.00	100.00	23.65	22.17	22.42	27.95
Sink in 2.96	66.14	15.28	33.68	7.04	8.76	1.19	91.26	15.13	17.90	94.05	21.57	3.36	4.02	26.48
Float on 2.96	33.86	7.83	6.44	77.16	78.40	0.16	8.74	84.87	82.10	5.95	2.08	18.81	18.40	1.47
−10-mesh (not treated)		4.47	21.74	38.60	40.36	0.78					4.05	5.39	5.40	5.88
Combined float and −10-mesh		36.77	7.57	74.54	75.63	0.24					11.18	84.84	83.03	11.76
Total sink		63.23	33.67	7.70	8.95	0.95					88.82	15.16	16.97	88.24

The data reported in Example 11 illustrate the benefits of heavy-media separation processes as applied to a siderite iron ore. The rejection of silica in both the minus 2 plus 1-in. and the minus 1-in. plus 10-mesh fractions treated in the cone is noteworthy. The combined sink products assayed only 7.70 per cent silica, and almost 89 per cent of the iron in the total ore was recovered in this combined product. Rejection of silica amounted to 85 per cent. Thus the role of heavy-media separation in this case was one of making a finished concentrate and a rejectable tailing in one operation.

Example 12.—*Topaz Ore*

Ore: Topaz ore from Southern United States
Mineralization: Fine-grained topaz associated with sandstone and a minor amount of gold
Analysis: Specific gravity, 2.98. Au, 0.029 oz. per ton. (Note: Topaz has a specific gravity of 3.5.)

Distribution of Values after Crushing to Minus 1 Inch

Product	Weight, Per Cent	Specific Gravity	Au, Oz. per Ton
−1 + ¼-in.	29.50	3.27	0.031
−¼-in. + 10-mesh	15.35	3.16	0.036
Primary −10-mesh[a]	44.72	2.72	0.025
Secondary −10-mesh	10.43	2.99	0.030
Total	100.00	2.98	0.029

Apparatus Used in Heavy-media Separation: 20-in semicontinuous cone
Medium Used: Ferrosilicon, minus 65 mesh; 60 per cent minus 325 mesh
Size Fraction Ore Treated: Minus 1 plus ¼-in. and minus ¼-in. plus 10-mesh

[a] "Primary" minus 10-mesh material refers to the screen undersize in the ore as received for testing. The "secondary" minus 10-mesh material refers to the screen undersize after crushing the original ore to minus 1 inch.

Results of Separation at 3.00 Specific Gravity, Minus 1 Plus ¼-inch Ore

Product	Weight, Per Cent: Of Size	Weight, Per Cent: Of Total	Specific Gravity
Original ore		100.00	2.98
−1 + ¼-in. cone feed	100.00	29.50	3.35
Sink in 3.00	73.41	21.66	3.51
Float on 3.00	26.59	7.84	2.89
−¼-in. + 10-mesh cone feed	100.00	15.35	3.22
Sink in 3.00	58.85	9.03	3.49
Float on 3.00	41.15	6.32	2.85
Primary −10-mesh		44.72	
Secondary −10-mesh		10.43	2.99

Tabling Test on Primary Minus 10-mesh Fraction of Ore

Minus 10-mesh ore was screened on 65 mesh. The minus 10 plus 65-mesh fraction was classified into three fractions, each of which was tabled to produce a concentrate, middling and tailing. The products from tabling of the three classified fractions were combined for analysis.

Tabling Test on Primary Minus 10-mesh Plus 65-mesh Ore

Product	Weight of Size, Per Cent	Weight of Original Ore, Per Cent	Specific Gravity
Table feed	100.00	44.72	2.72
Combined table concentrates	0.65	0.29	3.35
Combined table middlings	2.88	1.29	3.07
Combined table tailings	6.46	2.89	2.82
−65-mesh sands	73.88	33.04	2.67
−65-mesh slimes	16.13	7.21	2.84

Tabling Test on Secondary Minus 10-mesh Fraction of Ore

Procedure: The plus 10-mesh ore after crushing to minus 1-in. was screened to remove minus 10-mesh fines. The latter were wet-screened on a 48-mesh screen. The minus 10 plus 48-mesh fraction was classified into three products, each of which was tabled to produce a concentrate, middling and tailing. The latter products from tabling of the three classified fractions were analyzed separately.

Product	Weight of Table Feed, Per Cent	Weight of Secondary −10-mesh Fraction, Per Cent	Weight of Original Ore, Per Cent	Specific Gravity
Coarse table feed	100.00	19.84	2.07	3.26
Coarse table concentrates	30.39	6.02	0.63	3.43
Coarse table middlings	25.11	4.99	0.52	3.37
Coarse table tailings	44.50	8.83	0.92	3.08
Medium table feed	100.00	13.29	1.39	3.04
Medium table concentrates	29.52	3.92	0.41	3.35
Medium table middlings	27.06	3.60	0.38	3.14
Medium table tailings	43.42	5.77	0.60	2.79
Fine table feed	100.00	19.18	2.00	2.97
Fine table concentrates	10.59	2.03	0.21	3.21
Fine table middlings	17.94	3.44	0.36	3.23
Fine table tailings	71.47	13.71	1.43	2.87
Combined table feeds		52.31	5.46	3.10
−48-mesh fraction		47.69	4.97	2.80
Combined secondary −10-mesh fraction		100.00	10.43	2.95
Combined table concentrates		11.97	1.25	3.36

Example 12 shows how heavy-media separation processes may be applied for the concentration of unusual minerals, such as topaz. In this particular case approximately one half of the ore was minus 10-mesh in size and therefore was not treated by heavy-media separation. On the other half of the ore, approximately 70 per cent by weight of the cone feeds, or 31 per cent of the original ore, was recovered as a high-grade topaz sink product. An additional recovery of topaz, amounting to approximately 1.5 per cent, by weight, of the original ore, was obtained by tabling, but the grade of the table concentrate was not nearly as high as the sink product from heavy-media separation at a specific gravity of 3.00.

EXAMPLE 13.—*Tin-tungsten Ore*

Ore: Low-grade tin-tungsten dump ore from England
Mineralization: Cassiterite and wolframite with quartz, feldspar, muscovite and biotite
Analysis: Sn, 0.30 per cent; WO_3, 0.20

Distribution of Values after Crushing to Minus 1 Inch

Product	Weight, Per Cent	Assays, Per Cent		Distribution, Per Cent	
		Sn	WO_3	Sn	WO_3
Original ore	100.00	0.30	0.20	100.00	100.00
−1-in. + 10-mesh	74.98	0.23	0.15	57.76	55.56
−10-mesh	25.02	0.51	0.35	42.28	44.44

Apparatus Used in Heavy-media Separation: 20-in. continuous cone
Medium Used: Ferrosilicon, minus 65-mesh; 50 per cent minus 325 mesh
Size Fraction Ore Treated: Minus 1-in. plus 10-mesh

Results of Heavy-media Separation

Product	Weight, Per Cent		Assays, Per Cent		Distribution, Per Cent			
					In Size		In Total	
	Of Size	Of Total	Sn	WO_3	Sn	WO_3	Sn	WO_3
Results of Separation at 2.60 Specific Gravity								
Original ore		100.00	0.30	0.20			100.00	100.00
Cone feed	100.00	74.98	0.23	0.15	100.00	100.00	57.76	55.56
Sink in 2.60	61.64	46.22	0.33	0.23	86.75	97.30	50.16	54.05
Float on 2.60	38.36	28.76	0.08	0.01	13.25	2.70	7.60	1.51
−10-mesh (not treated)		25.02	0.51	0.35			42.24	44.44
Results of Separation at 2.70 Specific Gravity								
Original ore		100.00	0.30	0.20			100.00	100.00
Cone feed	100.00	74.98	0.23	0.15	100.00	100.00	57.76	55.56
Sink in 2.70	53.75	40.30	0.36	0.26	82.48	95.95	47.52	53.54
Float on 2.70	46.25	34.68	0.09	0.01	17.52	4.05	10.24	2.02
−10-mesh (not treated)		25.02	0.51	0.35			42.24	44.44
Results of Separation at 2.80 Specific Gravity								
Original ore		100.00	0.30	0.20			100.00	100.00
Cone feed	100.00	74.98	0.23	0.15	100.00	100.00	57.76	55.56
Sink in 2.80	7.07	5.30	2.06	1.74	62.40	83.11	35.97	46.47
Float on 2.80	92.93	69.68	0.09	0.03	37.60	16.89	21.79	9.09
−10-mesh (not treated)		25.02	0.51	0.35			42.24	44.44
Results of Separation at 3.00 Specific Gravity								
Original ore		100.00	0.30	0.20			100.00	100.00
Cone feed	100.00	74.98	0.23	0.15	100.00	100.00	57.76	55.56
Sink in 3.00	1.78	1.33	7.29	5.96	55.56	71.62	32.01	39.90
Float on 3.00	98.22	73.65	0.11	0.04	44.44	28.38	25.75	15.66
−10-mesh (not treated)		25.02	0.51	0.35			42.24	44.44

The results reported in Example 13 again illustrate the role of heavy-media separation processes as a preconcentration device in the treatment of low-grade dump ores. In this case, by subjecting the ore to heavy-media separation at 2.80 sp. gr., almost

93 per cent, by weight, of the minus 1-in. plus 10-mesh portion of the ore, corresponding to almost 70 per cent of the total ore, could be rejected in a discardable float product. The sink product contained 62.4 per cent of the tin and 83 per cent of the tungsten in the cone feed and would be combined with the minus 10-mesh ore for further treatment.

APPENDIX.—Apparatus and Technique, Semicommercial Tests

The 20-inch Laboratory Cone

The 20-in. continuous laboratory cone (Fig. 1) was developed at the Ore Dressing Laboratory of the American Cyanamid Co., in order that laboratory testing might be done under conditions comparable with those encountered in plant practice.

Description of Cone

The separatory cone *A* consists of a cylinder 20 in. in diameter by 6 in. deep, surmounting an inverted 70° cone of 20-in. base. This cone is truncated at a point where it is 2 in. in diameter and a flange is welded at that point (*B*). From the companion of that flange a 2-in. air-lift feed pipe slopes down at an angle of 60° to a point where a 150° bend will allow an air lift *C* to rise vertically outside the cone. The air lift rises to a point somewhat above the top of the cone and then discharges through an air-lift head *D* onto one side of a vibrating screen *F*, which is equally divided longitudinally. This longitudinally divided screen not only receives the discharge (sink) product of the air lift *C* but also the discharge (float) product from the weir *G* at the top of the cone. The medium that flows with these products is for the most part drained through screen *E* into the first compartment of the divided hopper *F*. The medium remaining on the products is washed by water sprays through the screen into the remaining compartment of the hopper. These water sprays are not shown but normally would occupy the position above *M*. The drained undiluted medium from the first compartment of the hopper is returned to the cone through the 2-in. air lift *H*. The highly diluted medium, which is washed from the sink and float products into the second compartment of the hopper *F*, is discharged from the hopper through pipe *I* into any suitable vessel placed below. The feed line to air lift *H* is made of standard 2-in. pipe and fittings, the angle made by the member *J* being 45°. The air-lift head *K* is of particular importance because it has an air escape at its top, which makes possible a comparatively quiet discharge of the air lift to the medium intake funnel *L* for the cone.

The medium intake funnel is welded as shown in Fig. 1 to the medium intake column *N*, which also serves as the shaft for a pair of sweeps, which continually scrape the side walls of the cone in order to prevent the settling of medium on the walls. The medium intake column extends almost to the bottom of the cone. This assembly of medium intake funnel, medium intake column and cone sweeps is rotated at the rate of 13 or 14 revolutions per minute by the small gearhead motor *O*.

Inside the medium intake funnel the medium intake column is perforated with two oval openings, which together are somewhat greater in area than the inside area of the 2-in. medium intake column. The medium intake column is further perforated below the funnel with ½-in. holes. These ½-in. holes are spaced evenly over the entire length of the column and are sufficient in number to make an aggregate area slightly less than the inside area of the 2-in. medium intake column. The medium intake column is also open at the bottom.

Air is supplied to the air lifts *C* and *H* through reducing valves *P*. The pressure through these air lifts is variable, dependent upon the requirements of particular problems. Generally, air pressures run between 5 and 10 lb. The reducing valves

are incorporated in the design because such low-pressure air is seldom available, and when high-pressure air is to be used the reducing valve will give much smoother operation than simple valve control.

The feed hopper occupies a position Q at the rear of the cone, but is not shown in Fig. 1. The hopper rests on a bracket fastened to the framework of the cone and is discharged into the cone by means of an electrically vibrated feeder. The feed rate of the feeder is variable up to a practical limit of about 500 lb. an hour.

Likewise not shown in Fig. 1 is a partitioned chute that is attached to the screen at R for conveyance of the products of the screen into receptacles placed on the platform S.

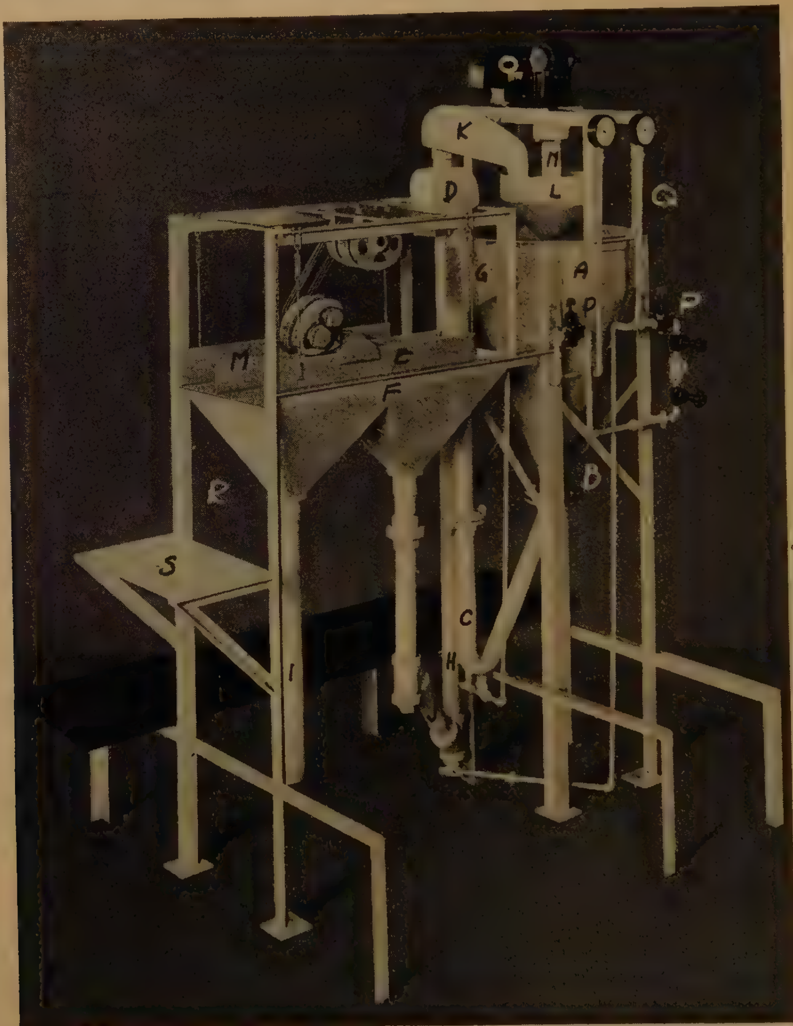

Fig. 1.

Media Used in the Cone

Various media may be used in the cone. The laboratory tests should be conducted

with the type of medium that is being used or will be used in the plant. Inasmuch as most plants using heavy-media separation processes use either ferrosilicon or a mixture of ferrosilicon and magnetite, these media are most used in the laboratory cone. In certain instances it is feasible to use magnetite alone. Galena is used in some plants.

Ferrosilicon.—Ferrosilicon commonly used in this process is designated as 15 per cent ferrosilicon, meaning a 15 per cent silicon content. This ferrosilicon may be obtained from the Electro-Metallurgical Sales Corporation, This company has two standard products, designated as Grade 80 and Grade 100. Grade 80 is relatively coarse and Grade 100 relatively fine. Either may be used in the laboratory cone, dependent somewhat upon the specific gravity desired in the medium suspension. In general, Grade 80 is used for working at high specific gravities and Grade 100 for working at low specific gravities. Either grade may be made finer by grinding in a ball mill if a special product is desired.

At the Ore Dressing Laboratory, two lots of ferrosilicon, coarse and fine, are used. The finer of these two lots is on the order of Grade 100; the coarse lot is similar to Grade 80. Typical screen analyses are given in Table 1.

TABLE 1.—*Screen Analyses of Ferrosilicon Media*

Mesh Tyler	Grade 80, Per Cent		Grade 100, Per Cent	
	Weight	Cumulative Weight	Weight	Cumulative Weight
+65	1.41	1.41	2.42	2.42
+100	8.65	10.06	4.08	6.50
+150	5.89	15.95	4.23	10.73
+200	9.72	25.67	9.16	19.89
+325	15.35	41.02	14.88	34.77
−325	58.98	58.98	65.23	65.23

The specific gravity of ferrosilicon is 6.7 to 6.8. A rough rule in practice is that heavy media may be used in suspensions having specific gravities up to half of the specific gravity of the solid material. Actually, in practice it is possible to use ferrosilicon suspensions having specific gravities somewhat greater than 3.4, or half the specific gravity of ferrosilicon. If the medium is clean a specific gravity of around 3.47 may be maintained without excessive difficulty, but most operators consider a specific gravity greater than 3.4 as difficult to maintain.

Magnetite.—Magnetite may be used either alone or in conjunction with ferrosilicon. It is best suited for use by itself when a medium of low specific gravity is desired. The specific gravities of magnetites available on the market vary considerably and care should be exercised to obtain a magnetite of high specific gravity and consequent high degree of purity.

Galena.—Galena has some disadvantages as a medium. Its principal disadvantage is the difficulty of cleaning it. This will be further discussed under a separate heading. Galena has a higher specific gravity (7.4 to 7.6) than either ferrosilicon (6.7 to 6.8) or magnetite (5.0 to 5.2), and might be expected to admit of a higher working specific gravity than ferrosilicon or magnetite. The hardness (2.5 to 2.75) of galena is low, however, as compared with magnetite (5.5 to 6.5) or ferrosilicon (7.3 to 7.6). This low degree of hardness favors more rapid production of fines in the medium than for ferrosilicon or magnetite. This occurrence of fines tends to increase the viscosity of the medium. A partial screen analysis of galena

TABLE 2.—*Partial Screen Analysis of Galena Medium Taken from a Plant Operation*

Mesh Tyler	Weight, Per Cent	Cumulative Weight, Per Cent
+65	1.20	1.20
+100	0.84	2.04
+150	1.25	3.29
+200	2.14	5.43
+325	6.79	12.22
−325	87.78	100.00

medium as used in one plant is given in Table 2.

Preparation of Media.—Ferrosilicon and magnetite as obtained on the market contain some impurities and it is necessary to clean them before using them in the laboratory cone.

Ferrosilicon as purchased must be wet for several days before any attempt is made to clean it. The most practical means of wetting it is to make a slurry of it with water. The ferrosilicon in this slurry, if allowed to stand for several days, will become thoroughly wet and may be cleaned easily in a magnetic separator. The simple expedient of pouring water into a drum of ferrosilicon is not satisfactory, as the water will not penetrate the bed of small particles and the material will not become wet. If ferrosilicon is cleaned before it becomes thoroughly wet a considerable loss of material will be unavoidable. Medium thus cleaned does not behave well in the cone. This is due probably to disproportionate losses in sizes during cleaning.

Magnetite does not require prewetting. A slurry is necessarily made in order that it may be fed to the magnetic cleaner but this slurry is made immediately before cleaning.

Galena must be cleaned by flotation or by tabling, which requires quite an installation of equipment. Galena, therefore, is not a very practical medium for laboratory work unless the laboratory has access to a source of prepared clean medium, so that the cleaning operation in the laboratory may be dispensed with.

Cleaning Ferrosilicon and Magnetite

In the laboratory it is considered good practice to start each day with a clean lot of medium. Two types of magnetic separators are available for cleaning medium in the laboratory. One of these, manufactured by the Dings Magnetic Separator Co., is of the "Crockett" type and is equipped with a permanent magnet bed; the other one, manufactured by the Jeffrey Manufacturing Co., is of the "Steffensen" type, which in turn is a variation of the Gröndal or drum type, and is fitted with an electromagnet bed.

The permanent magnet of the Dings-Crockett separator is to be recommended where no direct current is available, but the control of the field strength and capacity of the Steffensen separator gives greater latitude in the type of material handled and allows for some additional control on the ultimate concentrate produced.

When the Steffensen separator is used, care must be taken that the field strength is not too great, for if it is nonmagnetic material will be trapped in the magnetic concentrate and little or no cleaning will result. A little experimenting will soon determine what field strength is best. The magnet should always be installed with an ammeter in the circuit, and in the absence of a variable resistance a lamp bank should be provided. A satisfactory form of lamp bank is eight sockets in parallel, with the whole group in series with the field coils of the separator. Lamps of 25, 50, 75 or 100 watts can be screwed into these sockets and, depending on the size and number, small current changes may be made. With the parallel-series arrangement any light except the last one can be screwed in or out without interrupting the circuit, but the final break should be made by a switch lest the socket of the last light be burned out when an attempt is made to open the circuit by screwing it out of its socket. To take the inductive surge of the magnet field, a 50-watt light should be connected permanently across the magnet leads. If this light is not used, there is great danger of the high induced voltage injuring the insulation of the field coils when the current is interrupted. Some plants may have available a small motor-generator set, ½ or 1 kw. capacity at 32 to 50 volts. Such a set is an ideal means of exciting the Steffensen separator provided there is some way to control the amount of current flowing.

When a generator is used, an ammeter must be in the circuit, and the preferred means of taking the current off the separator is to shut down the motor-generator set, since magnetic separator. The medium discharged from the magnetic separator is very high in water content and must be partly dewatered before it can be used. The

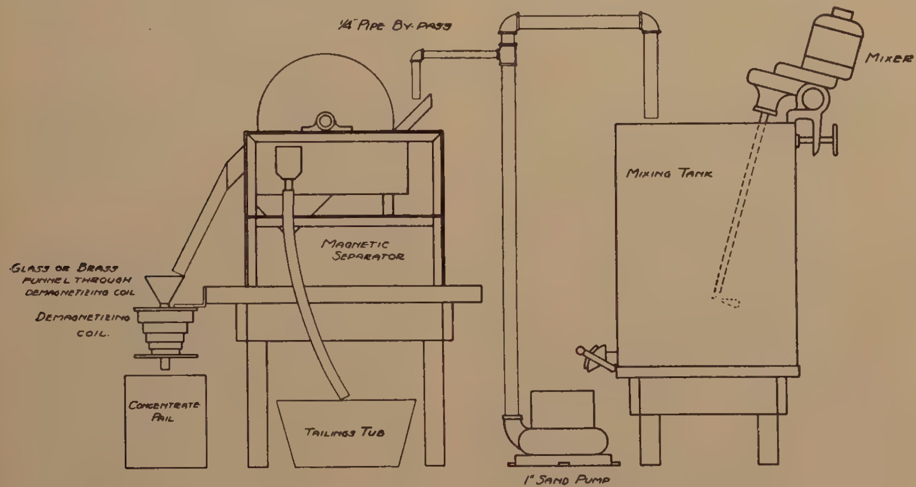

Fig. 2.

this method eliminates the need of a lamp to take the inductive "kick."

Wet ferrosilicon, or dry magnetite, new or dirty, may be cleaned as follows: Place 50 to 100 lb. of the medium in a 50-gal. drum containing about 40 gal. of water. This open drum should be equipped with a suitable stirring device, so that the medium is kept in complete suspension at all times. (Fig. 2 shows medium cleaning arrangement.) A molasses-gate valve near the bottom of the drum allows the medium suspension to be drawn off into a centrifugal pump. A screen at this point to remove coarse rock particles is desirable. The pump line carries the medium suspension up to and past the magnetic separator, where a portion is bled off for feed to the separator and the remainder is returned to the drum. A 1-in. pump is adequate and the feed to the separator may be bled off through a ¼-in. pipe from a tee in the pump line. The supply of ferrosilicon or magnetite in the drum may be replenished from time to time as the medium is drawn off through the medium must also pass through the demagnetizing coil before it is used. The medium may be passed first through an Akins-type spiral densifier, then through the demagnetizing coil, or it may pass directly through the demagnetizing coil and be dewatered by decantation from the receptacle into which it passes.

The demagnetizing coil is of utmost importance and no attempt should be made to operate without it. If the magnetized medium from the magnetic separator is not demagnetized before use in the cone, flocculation and consequent adverse change in the character of the medium will result.

It is best to have at least two batches of medium for the cone, in order that the cone may be operated every day if necessary, and with clean medium. Often a batch of medium will appear to be sufficiently clean for another day's operation before cleaning. Dirty medium has a high viscosity, and, generally, when the viscosity of a batch of medium starts to rise it increases rapidly. In the treatment of some ores, the second

day of operation on the same batch of medium may have to be stopped as a failure, and the test repeated with clean medium.

Charging the Cone

The first step in charging the cone is to open the valves to both air lifts, so that the first bucket of medium will start to circulate at once. The rakes are then started and the apparatus is ready to receive medium.

The medium is prepared for the cone by making a slurry with water. For making this slurry the use of a suitable stirring device with adjustable shaft is advisable. As each bucket of slurry is prepared it should be placed in the cone. After the first bucket of medium is placed in the cone it is desirable to work rapidly in order that an overflow may be obtained quickly and thus prevent the underflow from settling and thereby becoming too thick for easy circulation.

As soon as an overflow is obtained, the specific gravity of the medium should be determined. If the specific gravity of the medium is lower than desired, heavy, unslurried, medium should be added slowly until the desired specific gravity has been obtained. If the specific gravity of the medium is higher than desired, it can be lowered readily by addition of water. Regardless of whether medium or water is added in order to alter the specific gravity of the medium in the cone, an equal volume of material should be removed from the circuit. The cone operates best when it contains just enough medium to maintain an overflow when both air lifts are operating smoothly. If the circuit contains too much medium, the excess will remain relatively static in the hopper (Fig. 1) beneath the screen *E*. This will cause settling and consequent poor control of specific gravity in the circuit. It is best to charge the cone with slurry of greater specific gravity than is desired ultimately. The cone thus charged can be ready for operation in a minimum amount of time.

Specific gravities may be determined with any calibrated vessel and balance. At the Ore Dressing Laboratory a 1000-c.c. graduated cylinder and a beam balance are used. After the cylinder is tared on the balance, it is filled with medium from the cone and weighed, and the weight is read directly as the specific gravity. For example, a weight of 2850 grams net would indicate a specific gravity of 2.85. The cylinder should be filled from the top of the cone for the top specific gravities, with a small tin cup. The sample for determining the specific gravity of the medium at the top of the cone should not be taken directly, as it includes a water film and will give erroneous results. The water film should be gently splashed away on the surface of the medium and a cupful of medium should then be rapidly dipped. This should be repeated until the cylinder is filled. The specific gravity of medium from the bottom of the cone may be taken directly from the discharge of the air lift *C*.

Preparation of the Ore

For continuous operation of the cone, the ore should be crushed to pass 1-in. square-mesh screen. Some ores, of course, may require finer crushing in order to effect adequate liberation of the mineral. The crushed material should then be wet-screened in order to eliminate slime and other fine material. At the present time it appears that the second screening should be done at 10 mesh as the finest, thereby eliminating all minus 10-mesh material from the cone test. Slime will foul the medium and the fouled medium generally will become too viscous for use. If the second screening is done dry, the screened material should then be thoroughly washed. It is not necessary to dry the cone feed. The cone feed should be drained of all water that will drain off readily, but beyond draining no drying is necessary.

If a cone feed coarser than 1 in. is to be treated, the cone must be equipped either with larger air lifts or with a wire basket. This wire basket should be built of ¾-in. mesh screen and it should be so built that it will fit into the cone with no more than ½ in. of clearance at a level about 9 in. below the overflow weir. This basket should be fastened to the cone sweeps in such a manner that it may be installed or removed easily while the cone is filled with medium. Using this basket, the operator may treat 6-in. material or larger if he desires. However, he must remove the sink-and-float products by hand. The operator should remove the sink product frequently, because if it is allowed to accumulate in the basket it will stop circulation in the cone. This basket is not offered as one of the better features of the cone. The cone is designed for treatment of 1-in. material or finer. The basket, however, is a very satisfactory means of avoiding the laborious and unreliable alternative of bucket tests for small-scale testing of coarse material.

Operating the Cone

Good operation of the laboratory cone consists in running all tests of a series under the same conditions. Media used in two tests of a series should be as nearly the same as possible, even though the tests are made a week apart. This is easily accomplished by having enough of any type of medium to charge the cone twice and by starting each day with a clean charge.

Another main factor in cone operation is maintenance of the correct differential in specific gravity. A characteristic of the cone —in fact, one of the principles of heavy-media separation processes—is that the medium suspension at the bottom of the cone will have a higher specific gravity than the medium suspension at the top of the cone. By simply speeding up or slowing down the flow through the sink air lift *C*, the differential will be diminished or increased. In various other ways the differential in the cone may be altered. If a relatively large differential is desired, the holes in the side of the intake column should be plugged. If a moderate differential is desired, the medium intake column should have neither bottom hole nor side hole plugged. If a low differential is desired the bottom end of the intake column should be plugged. If the medium intake column is not plugged at its bottom end, it will supply a great part of the medium that goes through the sink-product air lift, and this will cause a static condition in the medium at the bottom of the cone and, consequently, in a relatively large differential. Plugging of the bottom of the medium intake column will cause the supply of medium for the sink-product air lift to come from the cone itself, and will cause a more rapid movement of the medium and consequent lower differential. Plugging of the side of the medium intake column accentuates the effect of operation with a medium intake column normal or unplugged. It should also be borne in mind that a finely ground medium will have a lower differential than a more coarsely ground medium.

When a series of tests is started, a flow through the air lift *C* adequate for the removal of sink product from the cone should be established. At the same time, a flow over the weir *G* adequate for removal of the float product from the cone, but not so great as to cause sink particles to be carried over the weir, should be established. When the flow through the air lift *C* is established, establishment of the flow over the weir *G* is mainly a matter of having the proper amount of medium in the cone. The flow over the weir should always be great enough so that the medium is actually overflowing, not simply the water film that generally is present at the top of the cone. After the air-lift and weir flows are established, the operator should determine the differential in the cone and in all future

tests of that series he should operate with that same differential.

The specific gravity of the medium suspension should be determined at frequent intervals. Certainly it should be determined before each separate test, however short the test may be. If a test is long, a rough determination may be made during the test. The feed need not be cut off for determination of the specific gravity during a test. The use of a wire screen cover for the cup will prevent interference of ore particles.

The air lift *C*, according to the law of passing particles, is too small for handling 1-in. particles. It is used partly because heavy-media separation is used on many ores that are crushed finer than 1 in. and partly because a larger air lift would be disproportionate to the size of the cone. Because the air lift is small, it will jam with 1-in. particles if it is fed too rapidly. If, for instance, the operator is treating an ore that contains a large fraction of sink material, he must govern his feed rate more or less according to the capacity of the sink air lift. If the ore contains a very small fraction of sink, the operator can use a greatly increased feed rate governed mainly by the speed at which the separation occurs in the cone.

In the event of stoppage of the sink air lift, the operator usually can clear it by closing the air-lift discharge, either by hand or with a stopper, causing the air pressure to force contents of the air-lift feed pipe back into the cone.

When all of the ore for a test has been fed to the cone, it will be necessary generally to wait 10 or 15 min. for the machine to clear itself before starting another test. After 10 or 15 min. have elapsed there may still be a few particles in the machine but most of these may be removed with a wire-mesh dipper. The few particles in teeter toward the bottom of the cone will not materially offset the results of the test.

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Heavy-media Separation Plant of the Barton Mines Corporation

By H. H. Vogel,* Member A.I.M.E.

(New York Meeting, February 1943)

This paper describes the milling practice and operating results of the recently installed heavy-media separation plant of Barton Mines Corporation, the world's largest producer of garnet. This pioneer application of heavy-media separation (sink-float) processes for the beneficiation of a nonmetallic ore is unique in many respects, the details of which will be apparent from the following presentation.

Characteristics of the Ore

The property is on Gore Mountain, 5 miles west of the village of North Creek, New York, and 11 miles by road. The altitude of the mine is approximately 2800 feet.

The valuable mineral, garnet, occurs in a surface deposit, the ore body being about ¾ mile long and varying in width from 50 to 300 ft. The garnet, principally almandite, $Fe_3Al_2(SiO_4)_3$, occurs in a metamorphic rock of uncertain origin. The gangue mineral is principally hornblende, which constitutes from 40 to 80 per cent of the rock. The remainder of the gangue is divided between plagioclase feldspars and hypersthene, with smaller amounts of biotite, apatite and pyrite. The garnet content of the ore body varies from 5 to 20 per cent of the mass and averages about 10 to 12 per cent. The garnet occurs as crystals, mostly imperfectly developed, known locally as "pockets." These crystals vary in size from a fraction of an inch to a foot or more in diameter. Occasionally crystals have been found up to 30 and 36 in. in diameter but they average 4 to 6 inches.

Nearly every crystal of garnet is surrounded by a rim of coarsely crystalline hornblende. The quarry faces are striking in appearance, showing crimson-red garnet crystals with their coal black rims scattered over a grayish black background. The specific gravity of the garnet is 3.8 to 4.1 while that of the hornblende ordinarily is 3.07 to 3.24. Some specimens of very dense hornblende have been found with a specific gravity as high as 3.40.

The garnet crystals are laminated in structure, and readily cleave into flat, tabular pieces with sharp, chisel-like edges. Even when crushed to a very fine size, the garnet retains this flat, sharp, slivery grain shape. This characteristic, together with its hardness and toughness, gives it its value as an abrasive.

Former Operations

Mining operations have been carried on for 60 years. Until 1923, only the oxidized (soft rock) section of the ore body was mined. The oxidation, which affected the garnet only slightly, made possible the removal of the garnet from the rock by hand after it was blasted down, using small hand picks. In 1923 a mill was erected on the property, and was rebuilt and redesigned in 1928.

Before the adoption of heavy-media separation, concentration was effected by

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jigging. The ore was reduced by means of a jaw crusher and a Symons cone crusher to pass a 1-in. round hole, and was screened on a ½-in. screen. The plus ½-in. material was reduced in rolls. The concentration of the garnet was accomplished by jigging in James jigs and in Hooper vanning jigs, the latter treating only the finest sizes. The slimes and minus $\frac{1}{64}$-in. hutch products were sent to waste without further treatment. As mentioned before, the garnet in this ore tends to break in very thin plates

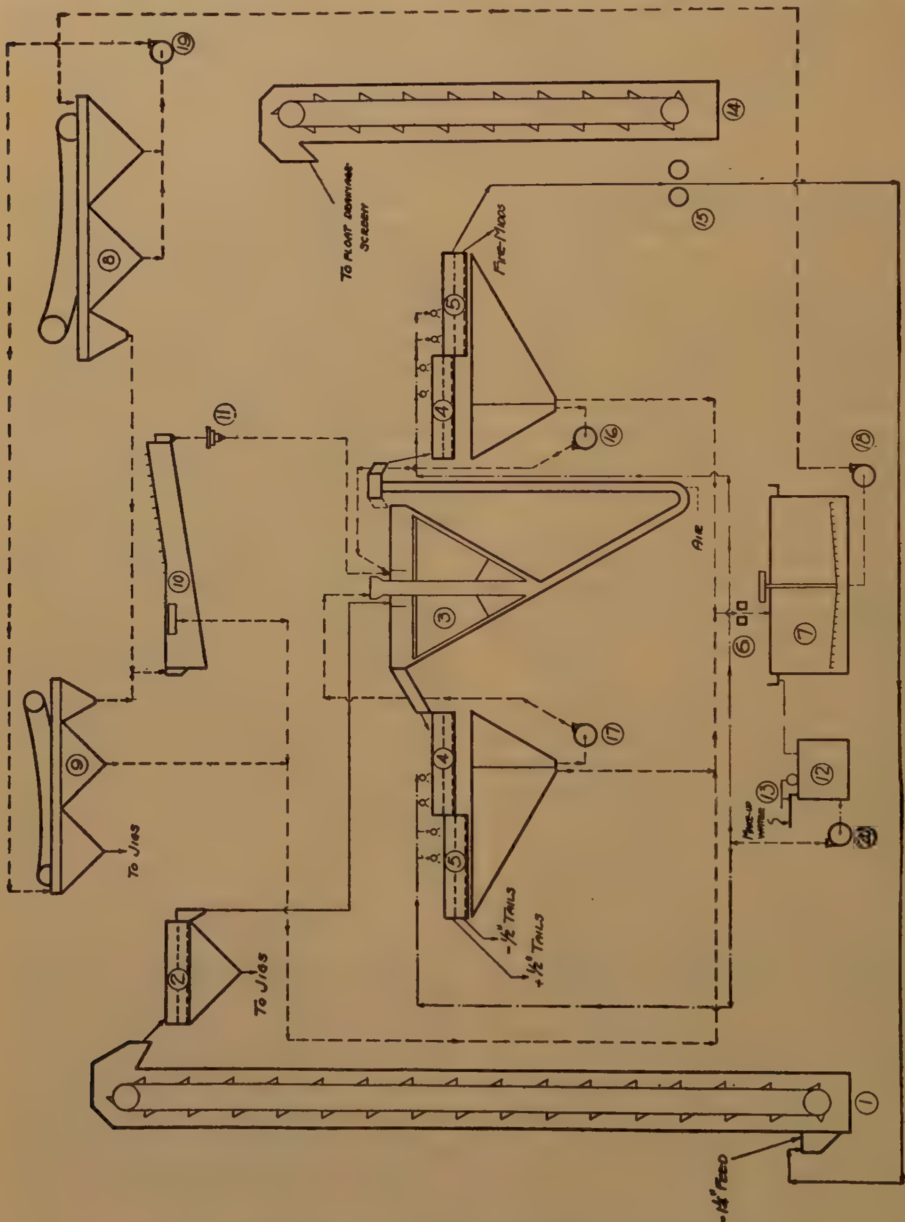

FIG. 1. FLOWSHEET OF HEAVY-MEDIA SEPARATION PLANT.

or slivers, while the hornblende breaks in chunky pieces. These characteristics made jigging difficult, especially on the finer sizes, as the thin garnet pieces tend to float off with the tailings. It is advantageous, therefore, to concentrate it as coarse as possible and also to avoid crowding the feed to the jigs.

Preliminary Investigation of Heavy-media Separation

In view of the characteristics of this ore, and also because the general grade of ore from the quarry had by 1937 become less rich, thus entailing the handling of considerable lean ore in future quarry operations, the Barton Mines Corporation became interested in the possibilities of heavy-media separation. Small-scale and pilot-plant tests were conducted during the winter months of 1937 at the Mascot Laboratory of the American Zinc, Lead and Smelting Co., and the results were very encouraging. However, in these tests ground galena was used as the medium and it was felt that considerable difficulty might be encountered in keeping the galena medium clean enough to maintain the gravity at the desired point. In addition, it was felt that the cleaning circuit necessary for reconditioning the galena medium was needlessly involved, as it would necessitate the installation of several thickeners, which, because of the rigorous climate, would have to be housed in heated buildings. This made the cost of the installation economically unattractive, therefore it was decided to await the development of a ferrosilicon medium that was being investigated by Butler Brothers.

After the management was satisfied of the suitability of ferrosilicon for the treatment of the ore, it decided to install a heavy-media separation plant. The plant was installed in a space formerly occupied by eight James jigs. The actual space requirements for the heavy-media plant, including all equipment, amounts to about 1000 sq. ft. It was placed in operation in September 1941.

Flowsheet and Equipment

The flowsheet of the process is shown in Fig. 1. The mill feed (minus 1¼-in. Symons cone-crusher discharge) is fed from the mill ore bin by a conveyor to the wet-mill feed elevator (1). This elevator discharges to a double-deck 4 by 10-ft. Ty-Rock screen (2) having ¼-in. square openings on the top deck and a No. 226 stainless-steel Ton-Cap on the bottom deck, with an opening of 0.116 in. The throughs of this screen go to a 6-ft. Allen cone for dewatering, then to a hydraulic classifier and then to the fine jig circuit.

The combined screen oversize is fed to the heavy-media cone 3. This feed constitutes 60 to 70 per cent of the total mill feed. The cone is 5 ft. in diameter with a 4-in. outside air lift. The gravity of the ferrosilicon medium in the cone is varied according to the character of the ore.

The top gravity of the cone is tested every 15 min., and gravities from 3.10 to 3.25 have been carried. The bottom gravity of the cone is tested each hour and usually runs five-hundredths higher.

The float-and-sink products discharge on two 4 by 10-ft. Ty-Rock screens, 4 and 5, in series. Each screen deck is divided by a center partition, to keep the products separate.

The bottom decks of both these screens are equipped with No. 921 stainless-steel Ton-Cap screens having an opening of 0.063 in. The top decks have screens of ¼-in. square opening, all except the last section of 3 ft. 4 in., which has a screen with ½-in. clear openings.

The last screen section separates the cone tailings and middlings into plus and minus ½-in. The plus ½-in. cone middling (sink) is fed to a set of 30 by 14 Traylor rolls, 15, and the crushed roll product goes back to the feed elevator, 1, and to the pri-

mary sizing screen 2, and so partly back to the cone.

The minus ½-in. middlings—that is, the product of the second screen surface—goes to the cleaner-jig circuit. All middlings from the cleaner jigs, after being recrushed in rod mills, also go back to the feed elevator and so over the primary screen, and so also partly back to the cone. This, of course, increases the feed to the cone over that of the original ore and also enriches it considerably.

The plus ½-in. cone tailings are carried by conveyor either to the tailing pile or to a steel tank for truck loading, as a considerable quantity of this material is sold for ballast.

The minus ½-in. cone tailings go to the mill-tailings elevator, where they join the tailings of the fine jigs. This material either goes to the tailings pile or to a tank for truck loading.

When the plant was designed, it was estimated that 5 ft. of screen length would be needed for medium drainage, 10 ft. for washing and 5 ft. for final draining. This has been proved to be excessive for this ore, and a single screen, 12 to 14 ft. long, would have been sufficient.

The sink-and-float medium drainage is kept separate. The sink drainage is pumped back to the cone by a 3-in. Wilfley pump, 16, and is discharged into the feed ring at the top of the cone. The float drainage is pumped back to the cone by a 3-in. Wilfley pump, 17, and is discharged near the bottom of the cone through rectangular openings cut in the hollow shaft that drives the cone rakes.

The medium that does not drain off the ore is washed off by a series of sprays, a movable tray installed on the drainage hoppers under the screen enabling the operator to bleed off some of the drained medium into the wash-water drainage, and so send variable amounts of medium to the cleaning circuit, as becomes necessary.

The wash water and the added portion of drained medium goes to the medium cleaning and conditioning circuit. It passes through a set of Alnico block magnets, 6, and then to a 12 by 7-ft. Denver thickener 7 equipped with a motorized raising and lowering devise for the rakes.

The underflow of the thickener goes to a 2-in. Wilfley pump 18 and is pumped to a 36-in. primary Crockett-type magnetic separator 8. The primary Crockett tailings are pumped by a 1-in. Wilfley pump 19 to a 12-in. secondary Crockett separator 9.

The magnetic products of both primary and secondary Crocketts discharge into a 24-in. Akins classifier, 10, equipped with a motorized raising and lowering device for the classifier rakes and also with a variable-speed drive. The Akins discharge passes through a demagnetizer 11 and is fed into the center feed ring at the top of the cone 3.

The classifier and Crockett overflows and the middlings from the 12-in. secondary Crockett are returned to the Denver settling tank. The tailings, or first spigot product, from the secondary Crockett go to the fine-jig circuit. The second spigot product from this Crockett goes to the thickener 7.

The overflow of thickener 7 goes to a steel tank, 12, of approximately 800 gal. capacity. The contents of this tank are pumped by a 2-in. centrifugal pump 20 to the spray-water circuit. This spray water is used on all except the last row of sprays on the washing screens and the water seal on the magnetic separators. The last spray on the washing screen and the sprays on the primary screen are connected to one of the dirty-water mill pumps. The lubricating sprays of the magnetic separators are connected to the clean-water mill lines. Should the circuit need more water than comes from these sources, it is added to the wash-water tank through a float valve 13. The clean-up elevator 14 takes care of all spills.

Operating Details and Results

The plant is arranged for operation by one man. Rate of ore feed can be varied by

the operator from the operating floor, which is level with the top of the cone. Medium is stored both in the thickener and in the Akins. Medium density can be varied in several ways: (1) by changing the speed of the Akins; (2) by raising or lowering the settling-tank rakes, or by raising or lowering the Akins rakes, thus varying the amount of medium in the circuit.

The latter gives the quicker change. For this reason, as much material is stored here as possible, especially over shutdown periods. As all of these adjustments are made by motorized devices, all can be controlled from the operating floor by push buttons.

Since the plant operates for 10 hr. per day, it was necessary to make provision for medium circulation during the shutdown period. After the feed is shut off and the ore is all out of the cone, the settling-tank rakes and Akins rakes are raised to their top position and the air-lift discharge spout is turned 180°, so that the cone medium is circulated continuously in closed circuit during the shutdown period.

The average consumption of ferrosilicon under normal operating conditions is just slightly over 0.4 lb. per ton of mill feed. The ferrosilicon used is preferably 100 per cent minus 65-mesh and 96 per cent minus 100-mesh.

Because of the mineral composition of this ore, chemical analyses of products produced in the operation are not a guide to metallurgical results. Tailings can be checked by visual observation. Samples are taken periodically and analyzed by screening and then hand picking the garnet middlings and tailings out of each size and weighing them.

Metallurgical results on the plus ½-in. cone tailings are nearly perfect. At no time has an analysis higher than 0.3 per cent garnet been found. The minus ½-in. tailing, while considerably better than the former jig tailings at this size, could be improved on and in all probability will be. The minus ½-in. cone tailings average about 3.0 per cent garnet on normal ore. On difficult concentrating ore they have run as high as 5.0 per cent.

As the cone tailings are split, about 50 per cent plus ½-in. and 50 per cent minus ½-in. total cone tailings average about 1.65 per cent on normal ore. This represents approximately 50 to 60 per cent of the total mill tailings.

This new plant is the first nonmetallic plant to adopt heavy-media separation (sink-float) processes, and it is also the first plant, other than those treating iron ore, to employ ferrosilicon as the separating medium. Another unusual feature is the high gravity of the suspension employed.

From the economic standpoint, the adoption of heavy-media separation by Barton Mines Corporation has resulted in increased recovery of garnet and savings in costs. An additional advantage of this new plant rests in the ability to handle considerably more tonnage with the existing equipment and without additional labor, by eliminating the greater part of the ore by a coarser grind than formerly was possible with straight jigging and after secondary crushing in rolls. Also, this process made possible the recovery of a greater percentage of the garnet in the range of minus ½-in. plus ⅛-in., and so cut down the loss in fine garnet on the fine jigs. An additional credit, although less important, comes from the sale of some of the coarse float product for use as road ballast.

Beneficiation of Scheelite Ores by Gravity Concentration

By E. H. Burdick,* Member A.I.M.E.

(New York Meeting, February 1942)

The difficulties inherent in table concentration operations as applied to gold, silver, lead and zinc ores, are accentuated in the scheelite mill, which has a flowsheet that is similar in general principles. The narrow "spread" in the specific gravity between scheelite and the usual gangue in which it occurs often makes its clean separation extremely difficult and in some cases impracticable. The problem is further complicated by the extreme friability of the mineral and its inordinate tendency to "slime," particularly where fine grinding is necessary and wet grinding methods are employed.

The physical aspect of the occurrence of scheelite in the ores, the kind and character of the rock in which the mineral is present, its hardness, friability and the specific gravity of its constituents, as well as like characteristics of the associated minerals—pyrite, pyrrhotite, sphalerite, and others, all have a bearing on the successful milling of crude ores and the production of a concentrate that is marketable with minimum price penalty or none at all.

Most tungsten deposits are considered low grade when the limiting factor of low percentage recovery by gravity methods is borne in mind. The great majority of deposits developed or in process of development carry from 0.4 to 1.0 per cent tungstic oxide. Ore carrying 1 per cent is considered good grade; 0.7 to 0.8 per cent, fair, and below that figure, low grade. Steady mill feed averaging upward of 1.0 per cent WO_3 is exceptional. Inasmuch as buying specifications demand a 60 per cent WO_3 content, the mineral content of the ores as mined must be concentrated to at least that degree.

A few companies have constructed chemical plants for the treatment of their complex concentrates and low-grade flotation product, but the smaller operator must rely on table concentration, borrowing the tools and to a large extent the methods of his predecessors in the gold, silver, lead and zinc mills. Whether these are the best tools and methods that can be devised is an open question. Except in rare instances, table concentration, however carefully conducted, does not yield a satisfactory final recovery of the mineral content in headings as indicated by chemical determinations of tungstic oxide content. A saving of 60 to 65 per cent is considered fair, and an average 80 per cent recovery, exceptional. Each scheelite ore body is a separate problem. Details of handling that give best results in one case may not be successfully operative at a property on the same contact zone and but a short distance removed. Supplementary treatment of tailings by flotation may be successful in producing a low-grade concentrate of around 15 per cent WO_3, which is adapted to further treatment chemically at a given property and entirely unsuccessful at another. This problem constitutes another of the unanswered questions.

The following paragraphs relate solely to the beneficiation of scheelite ores by table concentration and supplementary methods.

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Laboratory and Mill Tests

Experience has shown that results obtained by small-scale laboratory tests can be equaled or exceeded in full-scale commercial operations. The two groups of tabulations of results of such tests in Table 1 are indicative of those that formed the basis for the development of mill flowsheets, design of mills and their construction.

epidote type. For each, flowsheets were developed and mills constructed, which are now in operation. The plant treating No. 1 type has handled around 50,000 tons of ore and that treating No. 2 type, 2000 tons.

From laboratory tests and subsequent mill practice, the best ultimate extractions were attained by making a table concentrate carrying from 35 to 40 per cent WO_3. This product was raised to commercial grade by

Table 1.—*Average Laboratory Results on Extreme Types of Ore*

Character of Ore	Control Analysis, Per Cent	Roll-crushed to	Screen Analysis: Mesh	Screen Analysis: Per Cent	Table Treatment	Recovery, Per Cent
No. 1. Replacement in limestone; considerable garnet.	WO_3, 0.96 Fe, 9.0 Zn, 0.90 SiO_2, 31.4 CaO, 28.4 Al_2O_3, 4.68 S, 0.35	−20 mesh	−20+35 −35+65 −65	35 21 44 100	Entire amount over 4-ft. Wilfley table. Rejected tailings. Screened middlings over 35 mesh and retabled undersize with concentrates from first operation added. Reground +35-mesh middlings to −35-mesh and tabled	64.1
No. 2. Quartz-epidote-lime gangue. Scheelite coarsely crystalline. Individual masses ranging up to one inch in diameter.	WO_3, 2.12 WO_3 (calc.), 2.466	−16 mesh	−16+28 −28+48 −48	39.80 27.88 32.26 99.94	Each size group concentrated separately over 4-ft. Wilfley table	Mesh — Per Cent of WO_3 Contained −16+28... 80.29 −28+48... 93.44 −48... 78.23 Over all.... 82.81

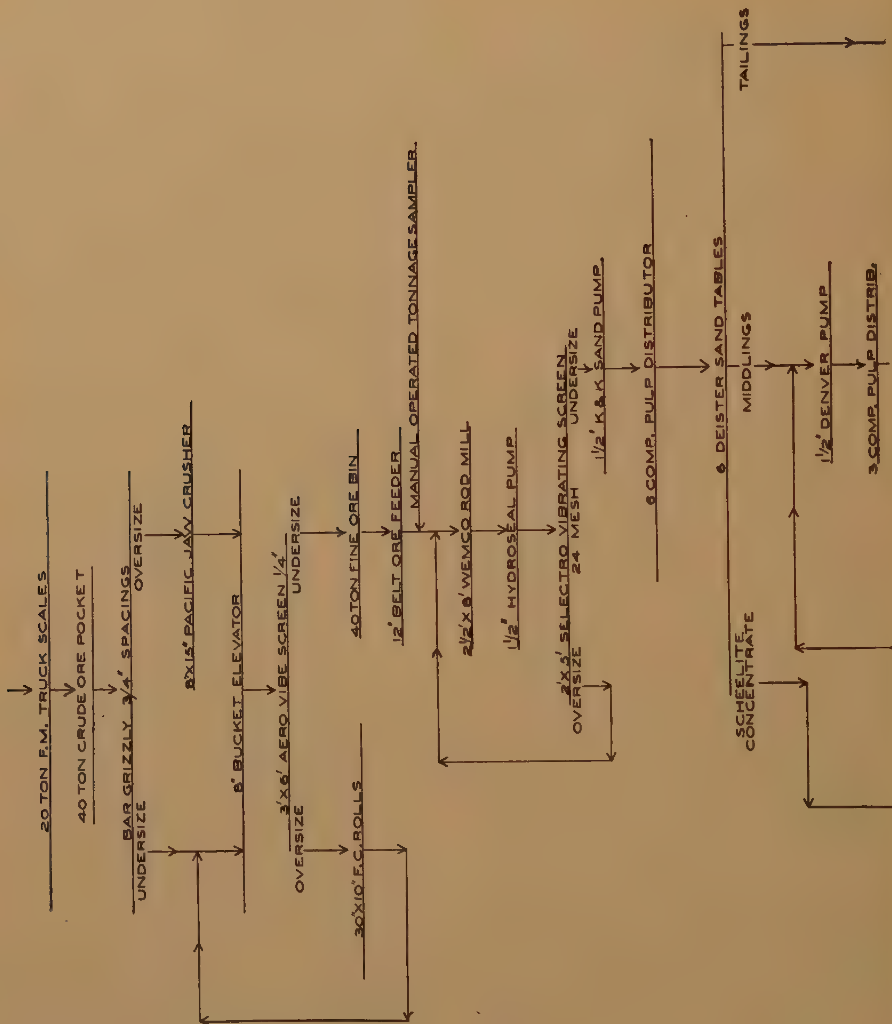
It has been demonstrated both in the laboratory and in commercial mills that the greatest mill loss is in slimes and fines.

As a general rule, the better the grade of mill feed, the better the extraction, up to certain maximum limits. An additional controlling factor is the size of the individual scheelite particles in the ore. If coarse enough to be freed at 16 to 20 mesh, higher extraction is attainable than with finer grind.

As indicative of the average laboratory results attained on two extreme types of ore, the figures of Table 1 are given. No. 1 represents tests on the garnet-lime type, and No. 2 the coarsely crystalline quartz-

subsequent retreatment by flotation or magnetic separation, or a combination of the two, as will be hereafter more fully discussed.

Mill tests on ores represented by No. 1 (Table 1) show results that are characteristic of replacement deposits in limestone that has been wholly or partly altered to garnet. Usually the scheelite is finely divided. In crushing and grinding in mill operations, the friable scheelite is reduced, not only by the action of the grinding unit, but also by the pebble action of the garnet fragments. These conditions produce an inordinate amount of slime tailings, which carry metal content in the ratio of 2:1 or more to sand tailings.


20 TON F.M. TRUCK SCALES
40 TON CRUDE ORE POCKET
BAR GRIZZLY 3/4" SPACINGS
OVERSIZE
UNDERSIZE
8"X15" PACIFIC JAW CRUSHER
8' BUCKET ELEVATOR
3'X6' AERO VIBE SCREEN 1/4"
UNDERSIZE
OVERSIZE
40 TON FINE ORE BIN
30"X10" F.C. ROLLS
12" BELT ORE FEEDER
MANUAL OPERATED TONNAGE SAMPLER
2 1/2'X 8' WEMCO ROD MILL
1 1/2" HYDROSEAL PUMP
2'X5' SELECTRO VIBRATING SCREEN
24 MESH
UNDERSIZE
OVERSIZE
1 1/2" K&K SAND PUMP
6 COMP. PULP DISTRIBUTOR
6 DEISTER SAND TABLES
TAILINGS
MIDDLINGS
SCHEELITE CONCENTRATE
1 1/2" DENVER PUMP
3 COMP. PULP DISTRIB.

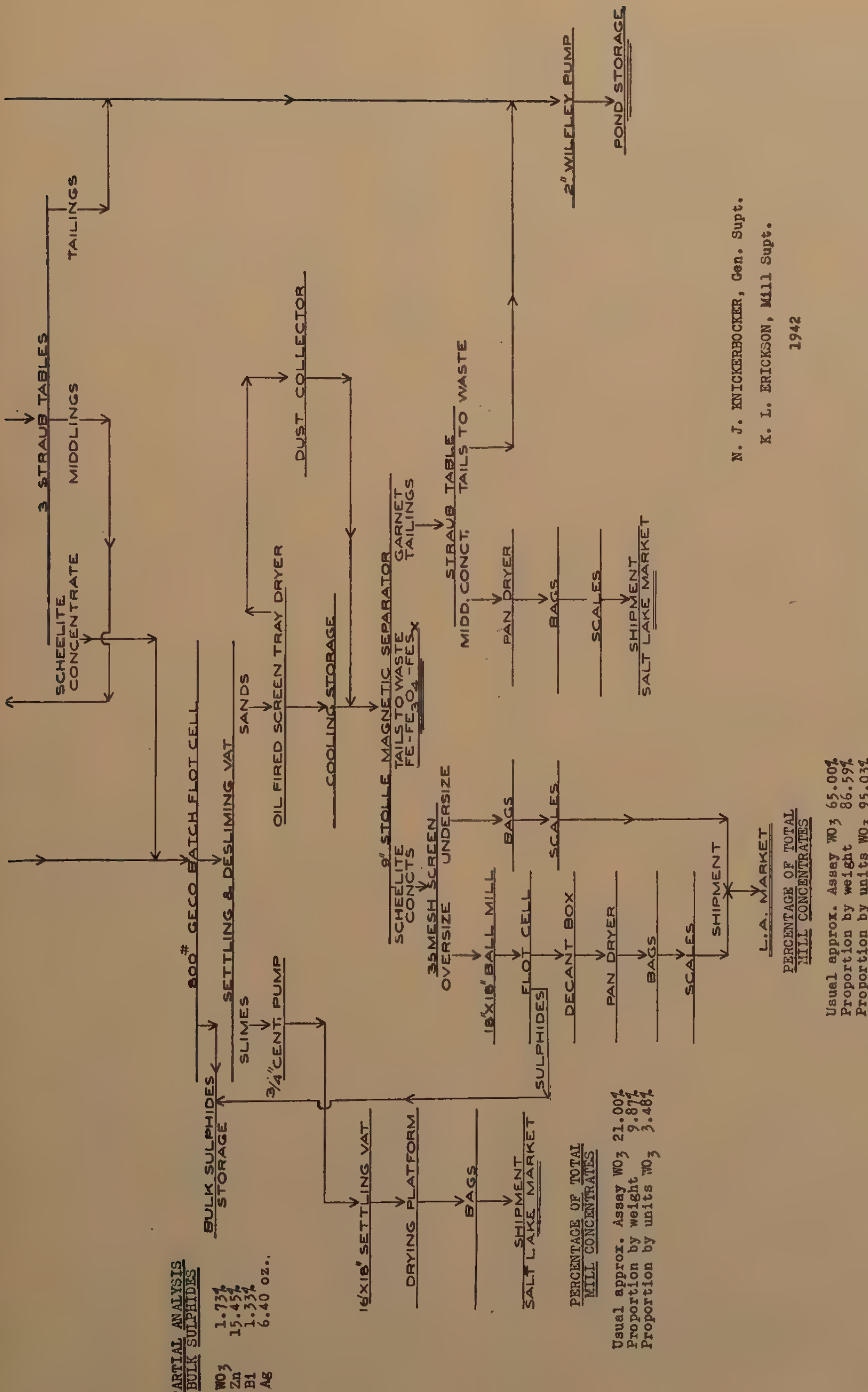


FIG. 1.—FLOWSHEET OF 75-TON GRAVITY CONCENTRATOR, LINCOLN MINES, INC.

Early in 1940, a table-concentration mill was built, following a flowsheet indicated by the laboratory tests. This plant had a capacity of around 40 tons per 24 hr., and has been in continuous operation. It has now been enlarged to a capacity of 80 tons and the mill processes have been considerably developed and refined. Average recovery on the same grade of ore has substantiated the preliminary laboratory tests, and the recoveries therein shown are being progressively exceeded.

The metallurgical problems developed by commercial operation over a period of 2½ years had been indicated by experimental work on these particular ores as well as on others of lime-garnet type.

Inasmuch as a good share of the following discussion is based on the operation of this mill, its graphic flowsheet is shown as Fig. 1.

The property is being operated by Lincoln Mines, Inc., and has treated to date around 50,000 tons of ore averaging slightly under one per cent WO_3. Finished concentrates produced, exclusive of by-products carrying tungstic oxide, have the following characteristic analysis: WO_3, 65.00 per cent; Mo, 0.32; Bi, 0.25; Cu, 0.04; P, 0.035; S, 0.70. Other deleterious elements are absent or under maximum allowable amounts.

Aside from the perennial recovery question and the many mechanical and metallurgical difficulties inherent in gravity concentration of any ore, experimental work and commercial operations developed two major problems that are peculiar to the scheelite industry, as follows:

1. The difficulty in obtaining checking analyses on low-grade scheelite ores and for those elements, sometimes present in minute amounts, whose determination involves the prerequisite of removing large percentages of tungsten before the determination can be made.

2. The removal from the concentrates of the elements necessary to meet trade specifications.

Concerning Analyses

In the summaries of analytical results hereafter tabulated, the three principal laboratories are designated by the letters *A*, *B* and *C*. Checking laboratories are designated as *X* and *Y*.

WO_3 *Determinations on Low-grade Material*

WO_3 determinations on low-grade headings and tailings are erratic in result. This is reflected in the irregular variance in indicated recovery percentages in Table 2, which covers the treatment of some 3000 tons of ore.

Table 2.—*Determinations on Low-grade Material*

Lot No.	Laboratory	WO_3, Per Cent		Indicated Extraction, Per Cent
		Heads	Tails	
1	B	0.573	0.055	90.4
	C	0.967	0.380	60.7
2	B	0.960	0.119	87.6
	C	0.810	0.241	70.2
3	B	0.891	0.183	79.4
	C	0.909	0.412	54.6

Average tailings assay on 6000 tons was 0.31 per cent WO_3 at laboratory B and 0.18 per cent at laboratory C.

Molybdenum Determinations

The usual maximum tolerance of molybdenum in trade specifications is 0.40 per cent. Analyses of five control samples yielded average results of 0.66 per cent at laboratory A and 1.11 per cent at laboratory B. Analyses at other laboratories check with neither. The situation in this regard has improved, although occasional cases of erratic results still arise.

Bismuth Determinations

The figures in Table 3 show extreme divergencies in determinations of bismuth. The average of control analyses on concentrate sales aggregating 50 tons was 0.37

TABLE 3.—*Bismuth Determinations*
PER CENT

Example	Lab. A	Lab. B	Lab. C	Lab. X	Lab. Y
1	0.23	3.10	0.25		
2	0.38	1.00		0.35	0.28

per cent at laboratory B and 0.22 per cent at laboratory A, the umpiring agency. When it is considered that the application of the former amount called for a price penalty and of the latter no penalty, the divergence is important.

Sulphur Determinations

No serious difficulty has been encountered in checking control determinations, although there is a consistent marked spread as between control assays and mill samples on the same material, indicating a difference in analytical methods in the two cases.

SALE OF CONCENTRATES

Concentrates from individual independent operations are largely sold to brokers, who act as intermediary purchasers or directly to ultimate consumers.

Trade specifications as to concentrate constituents vary somewhat with the contemplated use of the material.

All concentrates are sold on the number of units of 20 lb. of tungstic acid (WO_3) contained.

TRADE SPECIFICATIONS

The specifications in Table 4 represent a fairly liberal tolerance and a rigid requirement, respectively. The very low tolerance of various elements, even in the more liberal specifications, emphasizes the delicate nature of the required analytical procedure to determine them correctly. This work is greatly complicated by the necessity for removing the heavy tungsten content before such determinations can be made.

TABLE 4.—*Trade Specifications*
PER CENT

I		II	
WO_3....	60.00 minimum	WO_3...	70.00 minimum
Mo.....	0.40 maximum	Mo....	0.35 maximum
Cu.....	0.05 maximum	Cu.....	0.05 maximum
Bi.....	0.40 maximum	Bi.....	0.02 maximum
Sn......	0.10 maximum	P......	0.04 maximum
S.......	0.75 maximum	S.......	0.40 maximum
P.......	0.05 maximum		

CLEANING OF CONCENTRATES

The problems discussed in this paper are not those presented at the reduction plants equipped with chemical facilities for the complete beneficiation of even low-grade material. They are, however, some of the problems that arise at plants that are not large enough to carry the heavy capital expenditure necessary for the construction and operation of a chemical unit. These must rely on the resources available for proper preparation of their product for market.

Each type of table concentrates requires change in detail of treatment; however, the average main problems involved are similar in principle. Two general methods are usually followed:

Method 1

1. Extraction of the diluting, magnetic iron, garnet, epidote and other magnetic minerals by means of a Wetherell-type magnetic separator, operating under a high potential.

2. Quick magnetizing roast, coating the pyrite particles with a thin film of magnetic oxide.

3. Repass of the roasted material through the magnetic separator to remove the magnetized pyrite particles.

4. Sweet roast of the magnetically cleaned concentrates to remove, as far as possible, the remaining sulphur.

5. Return of the separator "cutouts" to the mill circuit for regrinding and reconcentration.

Method 2

1. Clean by flotation the table concentrates, removing the sulphides as nearly as possible.
2. Dry and remove the diluting magnetic minerals with a magnetic separator.
3. If necessary, sweet-roast the flotation-cleaned concentrates.

Problems

In most cases, the elements presenting the most serious problem are molybdenum, bismuth and sulphur. Of the three, the removal of molybdenum presents the greatest difficulty and, so far as known, is not extractable except by chemical treatment if present in the form of a chemical combination with tungsten and calcium (powellite type).

Molybdenum, if present as molybdenite, or associated with the iron minerals, is removable by table concentration, supplemented by flotation or magnetic cleaning.

Sulphur, if present as a sulphide, is removable by either magnetic, flotation, or roasting treatment, or a combination of the three. If as calcium sulphate, its removal, except chemically, is difficult or impossible.

Bismuth content can usually be reduced by proper flotation treatment (see Fig. 1). The effect of relatively high-temperature roasting has been observed by the writer in only one test involving a sufficient quantity of material to be persuasive.

Seven thousand pounds of concentrates carrying 0.40 per cent bismuth were sweet-roasted at temperatures up to 1200°F. The control determination on the roasted product was 0.05 per cent bismuth.

Magnetic "cutouts" in cleaning concentrates from garnet-bearing ores carried a substantial amount of WO_3, which necessitated their retreatment. This loss amounted to 10 per cent of the original values. Approximately half of this amount was recovered eventually by regrinding and reconcentration. Over-all recovery was approximately 95 per cent. Molybednum was not substantially reduced and bismuth was reduced approximately 50 per cent.

Magnetic separation with coarsely crystalline scheelite in quartz-epidote gangue is more efficient than other methods, and recoveries above 99 per cent are attainable.

Cleaning Concentrates by Flotation

Commercial tests of the efficiency of flotation treatment of high-sulphur table concentrates are now underway at the mill of Lincoln Mines, Inc., Hiko, Nevada. With recent changes in the ore body at the mine, maximum screen size has been changed from 35 to 20 mesh. Table concentrates containing 50 per cent sulphides or over are float-cleaned.

The following minerals have been recognized in the sulphides: pyrite, pyrrhotite, marcasite, sphalerite, bismuthinite, molybdenite.

The following facts have been developed as to the float susceptibility of the individual sulphides in the flotation procedure hereafter set forth:

1. Tarnished iron sulphides and all sulphide minerals of coarse size are most difficult to extract.
2. Coarse crystals of bismuthinite are only weakly activated, with resultant poor extraction. There is a need for a specific activator for this mineral.
3. Sphalerite appears to be the most completely extracted mineral.

Procedure

With a batch cell charge of 800 lb., the following reagents are added: Z-6 Xanthate, 2000 ml.; $CuSO_4$, 240 grams; pine oil (Yarmouth F), 200 ml. After conditioning for 5 to 15 min., the froth is pulled clear and clean. The air is then dropped and further reagents added: $CuSO_4$, 50 grams; Z-6, 500 ml. It is conditioned 1 to 5 min. and the froth pulled clear and clean. The air is again dropped and 500 ml. H_2SO_4 (coml.) is added. After a very short conditioning

period, the final froth is pulled as clean as possible.*

The dried, float-cleaned concentrates are screened over 20 mesh. The oversize is reground in a small ball mill and returned to the flotation cell.

While this procedure is not as yet completely standardized, it is giving satisfactory results, the last 18,000 lb. of concentrates having the following control analysis: WO_3, 65.60 per cent; Mo, 0.27; Cu, 0.04; Bi, 0.30; P, 0.02; S, 0.45.

Conclusions

Research work, confirmed by experience in a gravity concentration mill, dictates the following conclusions:

1. There is an urgent necessity for the standardization of laboratory chemical methods for the correct determination of small amounts of tungsten and associated elements in ores and products.

2. Average primary recovery on low-grade ores requiring grinding to minus 35-mesh will not exceed 70 per cent and often will be lower, depending on the grinding required to free the tungsten minerals.

3. Average recovery on coarse scheelite ores of from 1.5 to 3 per cent, and requiring grinding to minus 16-mesh will not exceed 80 per cent, the recovery factor being modified by the character of the ore gangue.

4. In all scheelite ores, and particularly those of garnet base, the slime and extreme fines loss in table concentration accounts for at least twice the loss in the coarser sand tails.

5. The weathering of tailings piles tends to free scheelite particles that remain attached to gangue fragments in the original mill operation, and, in some cases at least, additional economic recovery can be made by reconcentration of this material.

6. The methods used in the fine-grinding operations have a marked influence on recoveries that can be attained, and roll crushing, wherever adaptable, produces materially less ultimate sliming and extreme fines than ball-mill or rod-mill reduction to the same maximum mesh.

7. The recoveries now attainable by gravity methods preclude the successful development of large tungsten reserves and if average recoveries could be increased to 85 to 90 per cent, tungsten production would rest on a more solid footing and take a permanent place in the mining industry. (It is the writer's opinion that with a price range from a minimum of $25 per unit, and with 85 to 90 per cent recovery, a reliable domestic supply would be made available to meet our country's needs.)

Suggested Research

As bearing on the prime factor of recovery, research in the following fields is suggested:

1. Chemical laboratory practice.
2. Methods of grinding to produce a minimum of fine material.
3. Methods for positive classification by size of the concentrating table feed.
4. Modification, if any, of the mechanical features of gravity tables necessary to best accomplish increased recovery.
5. Secondary treatment of both slime and sand tailings.

While extensive research under these headings has been conducted by individuals and concerns as applying to their particular problems, such work does not necessarily directly apply to the widely scattered producing units, each with the same basic difficulty, that of recovery, and each with its local modifying factors.

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The Mechanism of Jigging

By Arthur F. Taggart,* Member A.I.M.E.

(New York Meeting, February 1943)

Recent jig practice has shown such marked departures from the pronouncements of the textbooks, particularly as to particle size recovered and size range of feed, as to make it desirable to reexamine the mechanism of jig action. This paper presents the results of such a study.

The general conclusions of the paper are:

1. A jig bed is a bi-functioned as well as bi-parted body. The upper layer comprises a plastic stream moving slowly in a substantially horizontal direction from feed lip to tail board. It serves to rough out tailing and light middling and to pick up heavy middling and lift it out of the jig box. The lower layer acts as the ultimate separating means by absorbing concentrate and rejecting heavy middling.

2. The separating function of the lower layer is performed in three definitely different ways, depending upon the relative sizes of the grains presented to it and those composing it.

Experimental

The experimental work consisted of close and long-continued visual observations of operating beds in a plunger jig, eccentric driven, and in determinations of the settling rates of various metal spheres through a quartz bed under various conditions of operation.

Falling velocities were measured by an ingenious method devised by A. C. Dorenfeld, a student in the Mining Department at Columbia. Spheres of different metallic alloys, ⅜-in. diameter, were cast with a short piece of fine copper wire frozen in to serve for suspension. A short piece of black cotton thread was looped to the top of the wire to serve as an indicator. The set-up for a test is shown in Fig. 1. The thread should be boiled, if necessary, to make it freely water absorbent. The piece of clean plate glass *D* is supported rigidly at the proper height in a vertical position. The white scale *E* is similarly supported behind the glass. At the beginning of a test, *D* is filmed with water by holding it (best with clean rubber finger cots) under running tap water. It is then clamped in approximate position and *C*, already wet, is laid across it and drawn straight and vertical with the upper end at or near the top of *D*, and the ball *A* slightly above the top of the expanded bed. The water film holds the string to the glass. For a test the string is released, its upper end position noted just after the top of the ball becomes submerged, when the corresponding time is read, and thereafter simultaneous readings of position and time are taken as desired.

Results of the visual studies of the bed are given at the pertinent places in the following discussion. Results of the settling tests are summarized in Figs. 2 and 3.

Definitions

The name *bed* as used herein refers to the entire mixture of solids and water filling the jig box. A *layer* is a stratum of a bed composed of particles of substantially the same specific gravities. A granular product is *long-range* when it approximates a crushed or ground product limited as to upper size only; a *short-range* product is one closely sized. A bed is *loaded* when operating with continuous feed and discharge of products; it is *unloaded* when pulsating normally but neither being fed nor

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discharging; it is *expanded* when loosened by pulsion, and *compacted* when the grains have all settled back into positions so that the jig box constitutes their entire support. The grains being treated are referred to as *sub-interstitial* when of such size that they can pass through the interstices of the bed without other than glancing contacts. *Super-interstitial* particles are those too large to penetrate the interstices without displacement of the bed particles. *Par-interstitial* grains penetrate the bed along interstitial passages without apparent displacement of the bed particles, but with constant scraping and turning, stopping each time the bed compacts; they are the particles ranging in size from the interstitial spacing in a fully expanded bed to something slightly larger than the spacing in a compacted bed.

Structure of Bed

A short-range bed of quartz and galena consists of two layers which are sharply distinct and separate, the separating surface being substantially horizontal. In an unloaded bed the grains in each layer range from larger below to smaller above for any given shape, and from equiaxed below to flats and slivers above for any given size. The spread by shape is more pronounced in the quartz layer than that by size, hence the top part of this layer is a mixture of large and small flats and slivers and of small, more rounded grains. The water passages therein are numerous, but small and highly tortuous. The bottom of the layer is composed of large equiaxed particles; it has relatively large direct water passages.

In a loaded bed the segregation in the quartz layer is not so distinct, not only because of the disturbance by the plunging feed and the horizontal flow of the layer, but also because of lack of time.

In a short-range bed of natural ore there is an intervening stratum comprising a multitude of thin layers and partial layers only roughly sorted. When such a bed is loaded there is definite intermixture of middling into the top layer and some penetration into the bottom layer.

Movement of Bed

The sequence of motions in an unloaded bed in a plunger jig during a cycle is illustrated diagrammatically in Fig. 4. Curve P

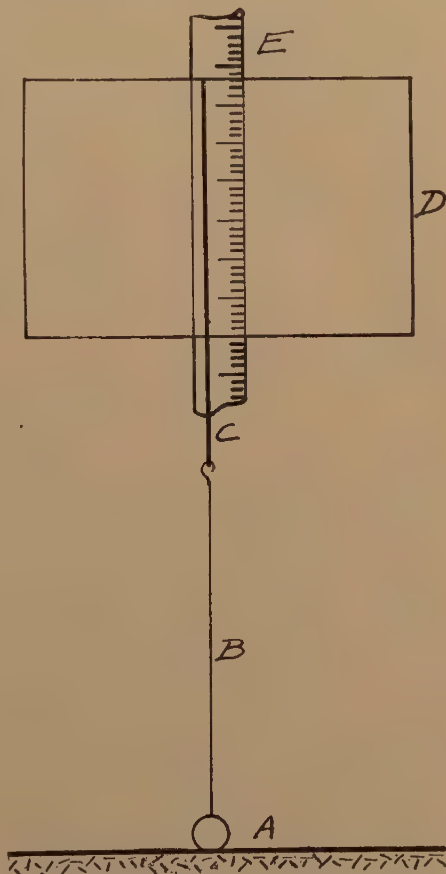

Fig. 1.—Apparatus for measuring rate of settling through a jig bed.
A, metal ball. D, clean glass.
B, copper wire. E, scale.
C, black cotton thread in position for start of test.

represents the motion of the plunger plotted as though it underlay the screen.

Curve W, representing water movement, shows the start of the water rise at a time t_1. The lag is due to slow return of water through the bed, loss over the tail board during the stroke, and open flap valves on

the jig plungers. The water starts upward abruptly at t_1, as the plunger meets the water surface in the plunger compartment, but the rate of rise (slope of curve) is less than that of the plunger because of leakage; the curve W falls steadily relatively to P and reverses at t_8 somewhat before the plunger has reached the bottom of its stroke. At the start of its fall, the water tends to follow the plunger closely, but as soon as the galena layer begins to settle and compact ($t_9 - t_{11}$), the flow is hindered increasingly until a minimum level is reached at t_{14}, whence level rises slowly with entry of plunger water until t_1'.

Curve Q_T is for the top of the quartz layer. The small and flat grains, responding quickly to interstitial water movement, start to lift at t_2. At t_3 the back pressure built up by interstitial friction causes the bottom of the galena bed to rise (curve G_B), lifting all above it, and imparting additional upward velocity to the top of the quartz by force transmitted through the as yet partially compacted layer. This motion continues, at a velocity increasingly below that of the water until at t_7 the settling velocity of the quartz relative to the water exceeds the rising velocity of the water, and the particles begin to fall. With reversal of water at t_8 the quartz grains are accelerated, the rate for an interval exceeding that of the water. During this interval, however, the top of the galena layer and, shortly thereafter, the bottom of the quartz layer, come to rest and consolidate and the resistance of the screen and side walls begins to build up through the quartz layer, hindering the

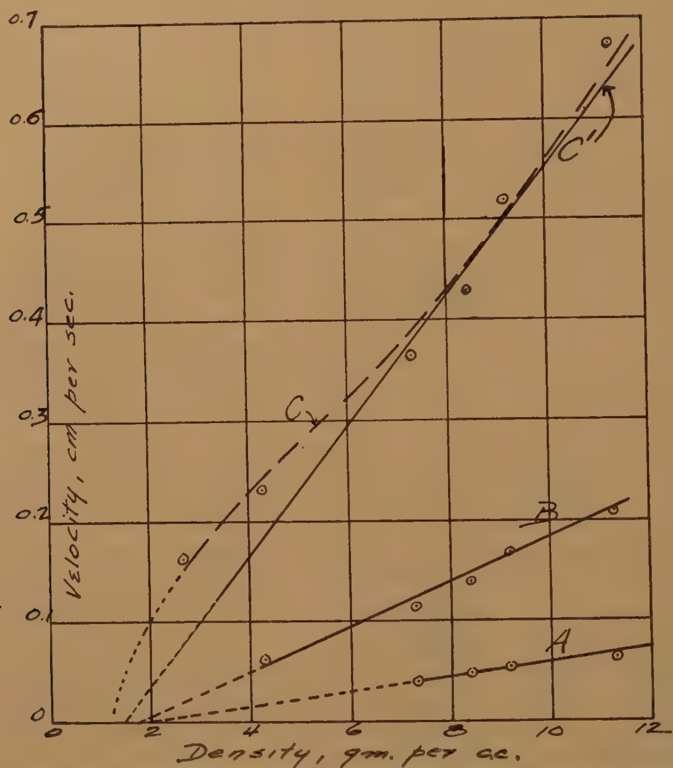

FIG. 2.—SETTLING VELOCITIES OF METAL SPHERES IN A 10 TO 14-MESH QUARTZ BED.

further fall of the top part, so that its falling rate decreases relative to that of the water. It finally comes to rest at t_{13}, remaining at rest until the succeeding cycle.

near to and in many cases before the time that the plunger reverses at t_{10}.

Expansion of the bed starts at the top of the quartz layer at t_2 and increases until

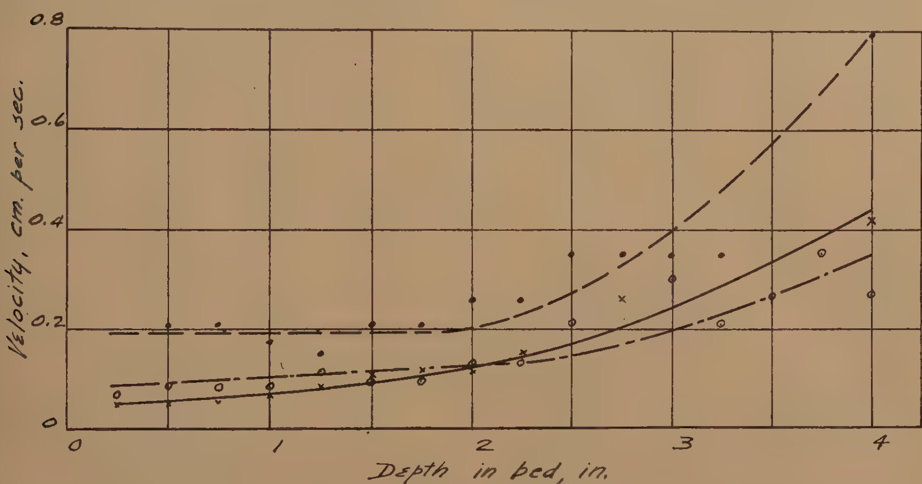

FIG. 3.—SETTLING VELOCITIES OF A METAL SPHERE (SPECIFIC GRAVITY, 9.2) AT DIFFERENT DEPTHS IN A 10 TO 14-MESH QUARTZ BED.

The bottom of the quartz layer (curve Q_B) is lifted *en masse*, as already noted, with the galena layer at t_3. As soon, however, as expansion of this part of the bed begins, the quartz grains are lifted faster than the majority of the galena grains directly below them (curve G_T) and the layers begin to spread apart. The quartz grains at the bottom of the quartz layer are larger and more nearly equiaxed than those at the top, rise at a lower rate and reverse earlier, at t_6. They begin to fall free-settling, accelerate at t_8 with the turn in water direction, and come to rest on top of the galena layer, already at rest, at t_{12}.

The top of the galena layer (curve G_T) has most of the movement characteristics of the bottom of the quartz, but comes to its top point sooner and lower down, falls somewhat more rapidly, and shows the characteristic hindrance due to compaction just before it comes to rest at t_{11}.

The bottom of the galena layer (curve G_B) rises most slowly, turns first, at t_4, and falls most rapidly, reaching the screen at t_9, the bottom of the galena layer comes to rest at t_9. Compaction occupies a longer time, not being complete until the top of quartz comes to rest at t_{13}. Expansion is not, however, uniform throughout. Visual impression that the top of the quartz layer is more compact than the bottom is confirmed by the settling rates shown in Fig. 3.

The grains in the lower half of a quartz bed rotate or oscillate very slowly around a mean axis so inclined as to approach the horizontal. Adjacent grains scrape each other lightly and give a strong visual impression that, at the instant of maximum expansion, they are in just-not-touching positions.

Optimum expansion in the upper layer is a function of the depth of that layer only. Stroke length and speed together determine lifting powers. Speed is limited by the fact that the bulk of the expansion takes place during the latter part of the pulsion stroke and the early part of the return stroke (see Fig. 4) and that the extent of expansion during the pulsion stroke decreases with the

sharpness of the impulse, which increases with speed. Increase in speed also decreases the time available for expansion by differential sedimentation. If pick-up starts before sedimentation is complete a tightened bed results. Increase in stroke length increases the time required for sedimentation, hence speed must be decreased as stroke length increases. Depth of bed, speed and stroke length must be so related that time is available for the stroke length that is required to give optimum expansion in a bed of the depth fixed by the height of tail board. Observation indicates that stroke length should be about three eighths the thickness of the upper layer of the bed.

Plasticity of Bed

Penetration was studied for particles ranging from sub-interstitial to super-interstitial dimensions several times those of the mean grain size of the bed, and for specific gravities varying from that of quartz to that of lead (11.3). Quantitative results on rates for super-interstitial spheres are given in Figs. 2 and 3.

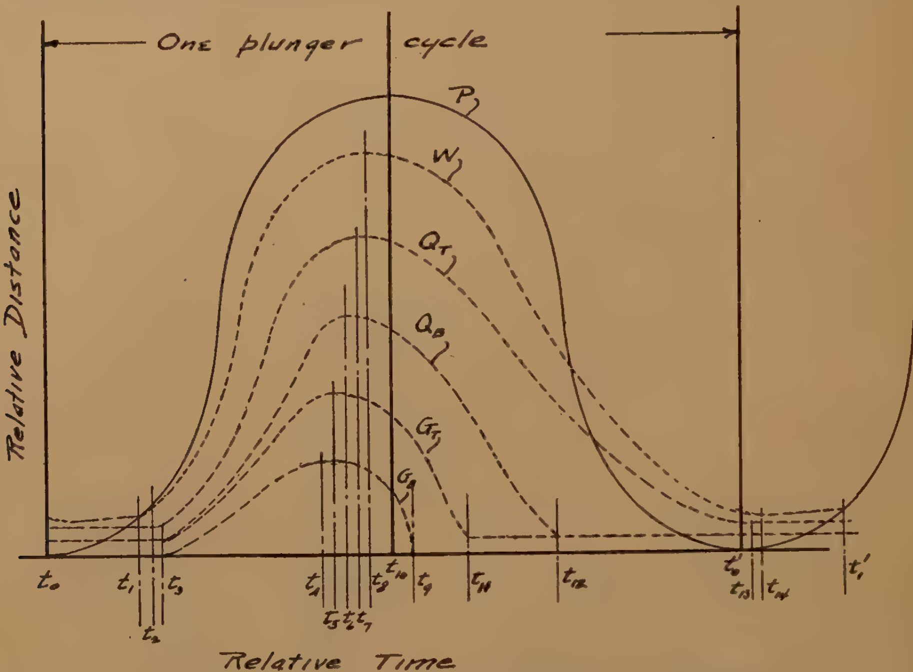

Fig. 4.—Diagrammatic analysis of movements in a plunger jig.

A quartz bed at rest will support a steel grinding ball without essential deformation. Working of the ball without downward pressure will, eventually, cause more or less penetration. It thus appears that resistance in this case is due largely to internal friction. Since flow of the layer has a definite threshold pressure, the phenomenon is properly designated plastic rather than viscous. In an expanded bed penetration by super-interstitial particles always involves crowding of adjacent bed particles into contact. Such particles, prior to sliding, must be subject to some threshold force. Again, the resistance is plastic.

Grain behavior in the neighborhood of a relatively large super-interstitial spherical particle penetrating an active bed is shown in Fig. 5. A generally random positioning of grains in the undisturbed bed is indi-

cated by the randomly directed lines farthest from the sphere. Directly in the path of the advancing sphere, the grains are definitely compacted. In compacting they arrange themselves with long axes roughly tangent to the surface of the sphere. As the sphere advances this compacted frontal sheath does not advance with it, but parts, the grains realigning to maintain tangency as indicated, while further compacting takes place at a lower level. A definite wake remains behind for a distance more or less equal to the sphere diameter, whereupon the wake grains again resume their random positions.

Large irregular particles penetrate point-down or edge-down and more rapidly than spherical particles of the same size and weight.

Thus when a large super-interstitial particle penetrates a bed it does work against the inner friction of the bed both in compacting and in sliding over and past the compacted particles. It also, of course, does work of upward displacement, since the material crowded from its path, both the visible grains described and the supporting water, are confined at the sides and bottom. This is the condition described by Stokes' law. The mobility equation for resistance of plastics to penetration is of the same general nature except for the addition of a negative constant to account for the threshold resistance, and the inner friction is defined as plastic resistance rather than viscosity.

In the experiments graphed in Fig. 2, the same quartz layer was used throughout and speed was constant at 250 strokes per min. The depth of the layer was 4 in. In the tests represented by curves *A* and *B* velocity was measured only for the top 1-in. part of the bed. For curve *C* the velocities are the averages for penetration of the top 3 in. Curve *A* is for ½-in. stroke, curves *B* and *C* for 1-inch.

Points for curves *A* and *B* fall, throughout their respective ranges, very close to straight lines. When straight lines are averaged through them and extrapolated, these cut the ordinate of zero velocity at sphere densities of about 1.85 and 1.95,

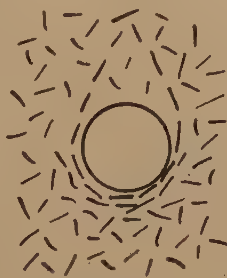

FIG. 5.—LARGE SPHERE SINKING THROUGH JIG BED.

respectively. Estimates of mean composite density of bed, based on the arithmetic average of the compacted and expanded volumes of the bed, were 1.74 and 1.85, respectively.

The linear character of curves *A* and *B* confirms the visual indication that the resistance of the quartz layer to penetration is due to inner friction in addition to the buoyant resistance. The extent of agreement between the values of sphere density for zero velocity, indicating the density of a sphere that would have no downward velocity in the bed tested, and the mean bed density, is remarkable considering the crudity of the methods of measurement. Fig. 3, showing increasingly higher velocities for a heavy sphere in the lower half of the bed, indicates less resistance there and a tendency to obey Newton's law, from which it may be concluded that either the plasticity or density of the bed is less in this region or that both have decreased. Since the particle crowding that causes plastic resistance is also a concomitant of increased density, the third alternative is the probable one. Visual examination, previously related, confirms the conclusion of greater average openness in the lower part. Hence the fact that the observed average density over 4 in. is somewhat lower than that indicated for the top inch

tends to support the results of the experimental measurements.

Curve C on Fig. 2 is more easily understood by comparing it with C', which is drawn through the mean of the experimental points for the four heavier spheres and the value for mean bed density on the zero ordinate. The discussion in the preceding paragraph shows that in settling through the upper 3 in. of the bed the spheres obey different laws in the upper and lower parts. The light spheres on curve C are more affected by diminution of plastic resistance than the heavy and hence depart further from C', the relative gain in velocity being greatest for the lightest sphere.

Particles of galena of bed size penetrating the quartz layer caused the same kind of disturbance as the larger spheres, although much less widespread, from which it is to be inferred that the nature of the resistance of the bed toward bed-size particles is the same as that toward the larger bodies. No acceptable method of measuring actual velocities was devised.

If corresponding values from Fig. 2 are substituted in Stokes' equation and the equation solved for frictional resistance, the values obtained are of the order of 2.1 to 2.3 poises. This is between 200 and 250 times the viscosity of water, or about that of a good cylinder oil at ordinary atmospheric temperatures. Adding something for the value of the constant in the mobility equation, the general order of agreement does not seem unreasonable. If the above argument is accepted, it follows, from experience with viscous liquids, that the plastic resistance of the upper layer of a jig bed is a large factor in determining its ability to carry along the reject of the lower or separating layer. The decrease in resistance in the lower part of this layer gives middling a better chance for presentation to the lower separating layer.

Plasticity of a galena layer was not determined. Such a layer, however, was more compact to visual inspection than the quartz layer.

Plasticity, once the threshold value is passed, does not prevent penetration of a layer by a particle of sufficient weight. It merely slows down the rate of penetration. For this reason it is effective in jig operation largely in supplying time factor in the upper or roughing layer, and in giving lifting capacity to this layer near the tail board when the rising velocity of the layer might otherwise be less than the falling velocity of a grain that it is desired to reject.

Density of Bed

The extrapolation of the curves of Fig. 2 to zero velocity indicated that the composite density of a bed layer is slightly less than that calculated from the mean volume of the layer. This is consistent with the indication of Fig. 4 that the rates of change of density are slower near the compacted end of the cycle than near the expanded end, thus allowing more than half of the total expanded interval for settling at rates corresponding to the lower densities and plasticities.

If it be assumed that the plastic resistance of a layer after settling has started is in accordance with the laws of viscous resistance to the extent that such resistance is zero at zero settling velocity, the layer densities obtained by extrapolation to zero velocity in Fig. 2 represent true mean densities. They correspond to 51.5 per cent voids at ½-in. stroke and 63.6 per cent voids at 1-in. stroke. The bed was too loose, judged by a millman's standards, with the longer stroke; about right with the shorter. If, on this basis, it is possible to conclude that the proper bed consistency is obtained when there are about 50 per cent voids, the proper density for any roughing (top) layer may be calculated readily.

If two particles—one of, say, galena; the other a heavy middling particle—both of

the same size, are conceived of as arriving simultaneously from the overlying light-mineral layer at a given point at the upper surface of a galena bed, they will, on the next stroke follow the time-distance path G_T, Fig. 4. The only difference between their behaviors, from a statistical standpoint, will be that the middling particle will rise very slightly above the maximum point of G_T, and not arrive at the compacted level until sometime between t_{11} and t_{12}.

But this slight difference in time will give the galena particle the first chance at the nearest available depression in the irregular compacting galena-layer surface. Once there, not only its greater weight but friction also will act to prevent its being pushed out by the middling particle when the latter arrives. Hence the middling particle will "float," as it were, on the bed while the galena particle becomes a part of it and thereafter "diffuses" slowly to the level therein prescribed by its shape and size (see Structure of Bed). The middling particle can penetrate only if it is large enough to displace a portion of a composite of galena and water that is lighter than itself, or if it is sub-interstitial. As far as bed-size grains are concerned, the effective density of a bed is that of the grains composing it. This is so because the internal friction, resulting from crowding by the supporting walls and bottom, prevents displacement of the lower particle by the downward impulse of the upper. In an operating jig the rejected middling particle is swept along the surface of the lower bed by the flowing upper bed, and is finally picked up at the tail board and swept out.

If the particle of lower specific gravity is considerably larger than the bottom-layer particles, the latter are at first swept from under it by the upward impulses and, being unable to return because of crowding, do not prevent it from falling into the hole thus made and penetrating a certain distance before compaction again occurs. As the particle penetrates, however, resistance to sweeping out increases by reason of the reaction of overlying grains, and if this resistance becomes sufficient before the top of the particle is completely submerged to prevent such sweeping, penetration ceases. At this time and in this position the buoyant effect on the large particle at full expansion is due to the excess of pressure on the under side of the penetrating particle exerted by the bed particles in suspension above that level, plus the water pressure due to the difference in head between the upper and lower surfaces of the particle. Prior to the instant of full expansion the force of the descending plunger is relieved only in part by flow through the small interstitial passages in the compacted bed, and pressure therein is at a maximum. As the bed compacts, and the suspended particles come into substantial contact again, frictional resistances increase and transmit to the penetrating particle the supporting influence of the jig box. The same analysis holds for particles of bed material or those of higher specific gravity larger than bed-grain size. Directly under and around the penetrating particle its motion and pressure compact the bed grains more than prevails elsewhere throughout the bed, so that here the frictional resistance disappears later on the pulsion stroke and reappears earlier. Also, in so far as these compacting particles are immobilized by the pressure upon them, they become a part of the penetrating body. But because of the space between them, their composite density may be less than that of the larger particle, in which case they act to increase its buoyancy in the looser part of the bed. Thus generally for particles of greater than bed-grain size the total resistance to the penetration is made up of a density factor and an internal-friction factor. These combine in different proportions and amounts throughout the cycle to give an *effective density* that ranges from a minimum corresponding substantially to the composite density at a particle

spacing of just-not-touching grains, to a resistance, wholly frictional, which is greatly in excess of the simple buoyant effect of a mobile liquid medium of a specific gravity far in excess of that any of the solids present in the system. Even a large gold ball will not penetrate a compacted galena bed, but it will readily penetrate mercury.

Since the effective densities are higher than the composite densities, the additional resistances must come from either the inner-friction effect already noticed, from energy transmitted from the jig mechanism, or from a transmitted reaction from the supporting box. There are no other upwardly directed forces of action or reaction in the system. Fundamentally, of course, the entire energy of the bed stems back to the jig mechanism, since it is the only source of energy input. On the other hand, without the reaction of the screen, all other things being equal, the entire feed would land in the hutch, unsorted. Hence this reaction must be considered a part of the force exerted by the bed to resist penetration. Interestingly enough, the plastic resistance of the bed is a maximum when the energy input is minimum. Hence, while this internal friction is a resistance to penetration, it is in opposite sense to the energy from the mechanism, decreasing as the latter increases.

When the bed material is sufficiently heavier than the grain that seeks to penetrate, so that at full expansion the effective density is greater than the density of the entering particle, that particle cannot penetrate at all, but floats throughout the cycle. What the density of such bed material must be to exclude particles of any given size can be estimated, if the percentage voids in a compacted bed thereof and the maximum volumetric expansion are known. Thus if x = decimal fraction of voids at full compaction, $1 - x$ = volumetric decimal fraction of solids. If y = the decimal fraction of expansion, $1 + y$ is the volume at full expansion. Then $\frac{1 - x}{1 + y}$ = the volumetric decimal fraction of solid in the expanded bed. If ∂ = specific gravity of the bed mineral and β = specific gravity of the mineral sought to be excluded.

$$\frac{1 - x}{1 + y}\partial + \frac{x + y}{1 + y} = \beta$$

Since this equation does not take frictional resistance into account, the bed-grain density may be slightly less than that calculated thereby.

Thus if the ragging of a jig has the density ∂ calculated by this equation, the concentrate will be free of super-interstitial and par-interstitial gangue, irrespective of the range in size of super-interstitial grains in the feed.

Time Factor in Jig Operation

The plasticity of the upper layer of the bed controls the time factor. If the rising rate at the tail board exceeds the settling rate of a particle in the bed, the latter will overflow. Hence, if plasticity prevents a particle from reaching the bottom of the upper layer before it reaches the tail board, it will overflow without ever having had a chance to try to enter the lower separating layer. This is the reason that increase in feed rate increases tailing assay and that decrease in feed rate may cause heavy middling to appear in cup concentrate.

Plasticity is controllable within limits by adjustment of stroke and speed. Within the operating range mobility of bed passes through a maximum at an intermediate point. Increased compaction is effected by both increase and decrease of speed from this point, stroke length remaining constant; at the optimum speed, decrease of stroke length causes progressively increased compaction, and increase usually causes first decreased compaction and thereafter irregular loosening; i.e., boiling.

Sub-interstitial Particles

Separation of particles that are completely sub-interstitial apparently depends

upon a combination of differences in free-settling rates in water and differences in resistance to scour by roughly horizontal water currents. The small particles pass through the interstices of a bed of relatively coarse ragging, partly in suspension and partly by being scoured along the upper surfaces of the individuals composing the ragging. The scouring and suspending forces are maxima at or shortly after the time t_1 (Fig. 4), when the rising current is approaching full velocity and the ragging has not yet begun to expand and enlarge the interstitial passages. The small particles in suspension at this time, which are predominantly the slower-settling, lighter particles, respond immediately, of course, to this current and begin to rise through the ragging. The light particles settled on the ragging, presenting greater surfaces in proportion to their weight than the heavier, likewise roll and leap and are the first of the settled material to go into suspension. The largest of the heavy small particles are rolled along on the supporting surface to a crevice and fall, never going into suspension at all. Hence throughout the period from t_1 to t_5 preferential rise of light particles is favored.

The falling interstitial currents of water are slower than the rising, as is evidenced by the fact that air collects under both the sieve and plunger of a plunger jig. The same conclusion can be reached, of course, by analysis from Fig. 4, bearing in mind that the returning force on the water is simply gravity, which is much less than the positive push of the plunger. Since the actual falling rates of the small particles are negligible by comparison with the interstitial water velocities, and the interference with fall and rise by the particles of ragging are substantially the same, the particles that remain in suspension and those that are readily scoured into suspension pass over the tail board with the overflow water. Substantially only those that resist suspension pass down through the ragging.

Par-interstitial Particles

These particles penetrate a bed without material disturbance of the surrounding particles, which is to say, with a minimum of plastic resistance. Observation of an operating bed shows that the heavy-mineral particles of this size, in penetrating the upper layer of the bed, release and begin to fall at the first sign of loosening of the bed proper on the pulsion stroke and are not lifted relative to their light-mineral neighbors at any part of the stroke. Exerting insufficient downward force to compact and push the light-mineral particles out of the way, they are slowed or stopped by them on each encounter during their fall, and thus settle more slowly than the bed-size heavy particles. On the other hand, they are more greatly interfered with during fall than the sub-interstitial particles, and tend, therefore, to settle more slowly than these. Since they penetrate by falling against the rising current during the time that the bed is expanded, their ratio of weight to cross section must be greater than that of the larger bed particles. Hence they must be of higher specific gravity. In other words, a bed excludes par-interstitial particles of its own or lower specific gravity and only admits those of sufficiently higher specific gravity to satisfy the weight-cross-section limitation stated above.

Other Types of Jigs

Action of beds in pulsator and in movable-sieve jigs was not observed. Analogy indicates that in a pulsion jig running with a closed hutch there is no runback of water such as is illustrated by the downward leg of curve W, Fig. 4, to the right of t_8. Rather, the water curve follows a course roughly similar to that of curve P from t_0 to t_{10} and then runs horizontally to t_0', while the valve is closed. The shape of the rising legs will vary with the type of valve. Without a special valve they may be taken, in general, to approximate those for the plunger jig.

The falling legs of the curves for the solids will all have smaller slopes, owing to lack of acceleration by the falling water.

The motion analysis for a movable-sieve jig must differ markedly from those of the plunger and pulsion jigs. Considering the Hancock jig, since the compacted bed is lifted by the screen, the slopes of the rising legs of the solids are the same as that of the screen. The water in the bed is lifted with the bed but percolates downward so that its slope is less and total rise is not as great as that of the solids.

The downward branches for the curves for all components are separate. The heavy screen frame, falling by gravity, probably accelerates throughout despite resistance by the water in the tank. In falling it drops from under the bed. The water, now lacking the upward impulse of the screen and the accompanying compacted bed particles, begins to slump downward to establish level with the water in the tank. To do so, however, it must pass through the screen. It is, therefore, hindered, and thus accelerates more slowly than does the screen. When the bed begins to compact again the water is further slowed increasingly until the end of the stroke.

The entire bed begins to fall directly the screen drops away from under it. Its fall is resisted by the column of water within the walls of the screen box to the extent that there is an excess differential between the falling velocity of the solid mass and that of the subsiding water. This is small at first. The only effects on the bed as a whole are a relative lifting of the top of the quartz layer and a fall of the bottom of the galena layer. With time this action continues to progress inward from both faces of the bed until maximum expansion occurs. Thereafter the bed compacts progressively from the bottom upward in the fashion described in detail for the plunger jig. Whether or not compaction of the top is hastened by lifting the remainder of the bed up to it, depends upon speed and length of stroke.

Summary

A new "theory" of jig operation is offered. The essentials are:

1. A jig bed consists of an upper roughing and transporting layer and a lower separating layer.

2. Plastic resistance is an important element in the functioning of both layers for particles of all sizes except sub-interstitial.

3. The effective or separating density of a bed layer as respects particles of bed size is that of the particles constituting the bed.

4. The effective density for particles larger than bed size is somewhat greater than the mean density of the bed (probably between 5 and 10 per cent).

5. Sub-interstitial particles are separated by a combination of simple free settling and of launder or streaming action.

DISCUSSION

(*T. B. Counselman presiding*)

F. C. Bond,* Milwaukee, Wis.—This study of the action in a jig bed quantifies features of the separation, and the conclusions reached as a result of measurements confirm in general the commonly held ideas of jig action.

It is noteworthy that no recommendations are made regarding closely sized feed versus mixed feed, and that the study did not include pulsion and suction strokes of unequal time duration, nor establish a relationship between particle size and length and duration of stroke.

The extrapolation of the settling velocities of spheres of different specific gravities to zero velocity establishes an "effective" density of the bed, and this density might be used to advantage in calculating hindered-settling ratios.

The most valuable part of the paper appears to be the discussion of the high internal friction within the jig bed, and the retarding effect of this high viscosity upon the rate of penetration and separation. The appropriateness of designating this high viscosity as plasticity may be questioned, since a jig bed has always been considered as having the properties of a liquid rather than those of a solid, but in any case it serves to emphasize the dominating factor of high internal friction.

* Allis-Chalmers Manufacturing Company.

Principles of Flotation, X—Influence of Cations on Air-mineral Contact in Presence of Collectors of the Xanthate Type

By Keith Leonard Sutherland*

(New York Meeting, February 1941)

This paper is a study of the differential flotation of the sulphide minerals in the presence of salts of silver, lead and zinc. In practice, accidental activation due to these salts is more important than their use as reagents to replace copper sulphate.

The general method of investigation has been based upon contact tests, the temperature of 35°C. being chosen in conformity with earlier work. Potassium ethyl xanthate and potassium amyl xanthate were selected as collectors and pure sulphide minerals were used. The experimental procedure has been outlined in earlier papers and all solutions were kept carbonate-free.[1,4] Contact, or spreading of an air bubble, at a mineral surface indicates the presence of an adsorbed collector film.

Purity of Reagents

The xanthates were purified in the usual manner.[1] Neither the cyanide nor the sodium hydroxide solutions contained sufficient carbonate to give a precipitate with barium chloride solution.

The silver sulphate was prepared from pure silver nitrate by evaporation with concentrated sulphuric acid until there was no evidence of brown fumes. This solution was diluted and the difficultly soluble sulphate washed with water so that unchanged nitrate was dissolved. The resultant crystalline powder was pure white.

The lead and zinc salts were prepared by dissolving especially pure lead and zinc in C.P. nitric and sulphuric acid solution and the crystals drained. The standard solutions were checked by titration. It was felt desirable to use the sulphate radical wherever possible, but since lead sulphate was insufficiently soluble for the purpose of the experiments, it was necessary to use some other lead salt.

A.R. copper sulphate was used without further purification.

Results

The results are presented in the accompanying figures; the areas of noncontact (nonflotation) are shaded in Figs. 8, 9, 10 and 11; in the remainder they lie to the right of the curve.

Zinc Sulphate

The curves in Fig. 1 are to be compared with those for Fig. 2, for which zinc sulphate is absent. This comparison shows that the influence of the zinc ion is usually small. Its influence is more marked, however, for chalcopyrite in the absence of cyanide. The influence of zinc sulphate is also marked for copper-activated sphalerite, for which tests were made only in the absence of cyanide. The results are of importance in the separation of pyrite from copper-activated sphalerite when zinc sulphate is added in the lead flotation section (p. 204 of ref. 1).

Sphalerite does not adsorb the collector (i.e., become floatable) under the conditions chosen.

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[1] References are at the end of the paper.

Lead Nitrate

The behavior of chalcopyrite, sphalerite and pyrite resembles that of galena (the

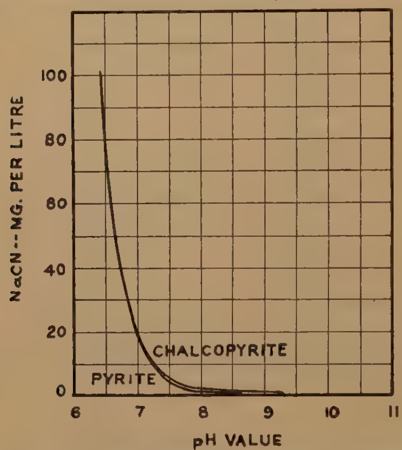

FIG. 1.—RELATIONSHIP BETWEEN CONCENTRATION OF SODIUM CYANIDE AND pH VALUE NECESSARY TO PREVENT CONTACT AT MINERAL SURFACES.

$ZnSO_4.7H_2O$, 173 mg. per liter; KEtX, 5 mg. per liter; Na_2CO_3, nil; temperature, 35°C.

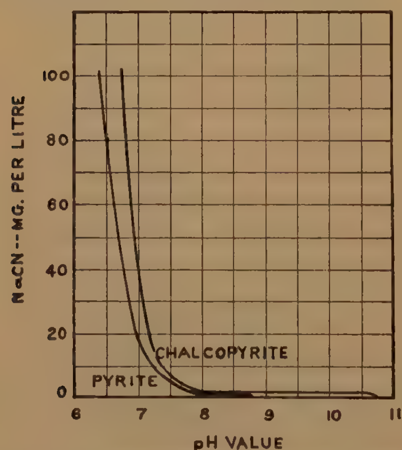

FIG. 2.—RELATIONSHIP BETWEEN CONCENTRATION OF SODIUM CYANIDE AND pH VALUE NECESSARY TO PREVENT CONTACT AT MINERAL SURFACES.

KEtX, 5 mg. per liter; Na_2CO_3, activator, nil; temperature, 35°C.

response of the mineral independent of cyanide as a depressant), but only the first two minerals behave quantitatively like galena (Figs. 3 and 4), pyrite requiring a lower pH value to adsorb the collector.

Covellite behaves differently from the other minerals in that the mineral surface

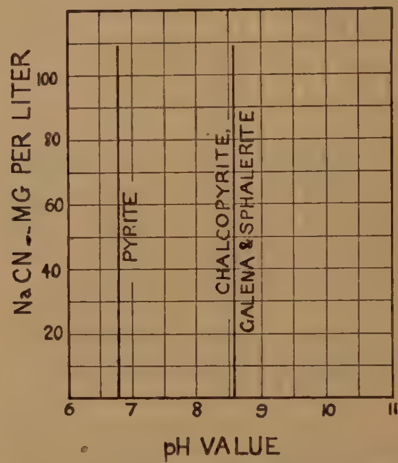

FIG. 3.—RELATIONSHIP BETWEEN CONCENTRATION OF SODIUM CYANIDE AND pH VALUE NECESSARY TO PREVENT CONTACT AT MINERAL SURFACES.

$Pb(NO_3)_2$, 199 mg. per liter; KEtX, 5 mg. per liter; Na_2CO_3, nil; temperature, 35°C.

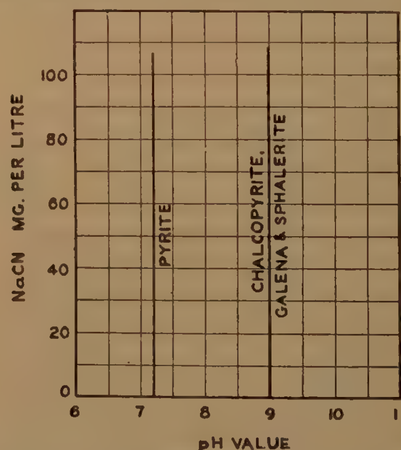

FIG. 4.—RELATIONSHIP BETWEEN CONCENTRATION OF SODIUM CYANIDE AND pH VALUE NECESSARY TO PREVENT CONTACT AT MINERAL SURFACES.

$Pb(NO_3)_2$, 199 mg. per liter; KEtX, 25 mg. per liter; Na_2CO_3, nil; temperature, 35°C.

is still amenable to depression by cyanide. Thus the curve showing the relationship between cyanide concentration and pH

value is of the type usually obtained in the absence of heavy metal salts. Fig. 5 compares the curve for covellite with that for sphalerite preactivated in 150 mg.

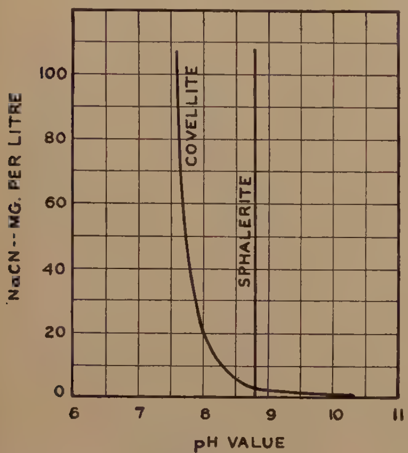

FIG. 5.—RELATIONSHIP BETWEEN CONCENTRATION OF SODIUM CYANIDE AND pH VALUE NECESSARY TO PREVENT CONTACT AT MINERAL SURFACES.

$Pb(NO_3)_2$, 199 mg. per liter; KEtX, 5 mg. per liter; Na_2CO_3, nil; temperature, 35°C.

$CuSO_4.5H_2O$ per liter solution, which coats the surface with a Cu-S film and which might have been expected to behave like covellite.

The critical pH value for galena in the absence of lead ions[5] at 35°C. is 10.4 and 9.8 in the presence of 25 mg. and 5 mg. potassium ethyl xanthate per liter, respectively. When an excess of sodium carbonate (150 mg. per liter) is added to precipitate the lead as carbonate from the lead-bearing solutions, the critical pH value is raised from 8.7 to 9.8 for solutions containing 5 mg. potassium ethyl xanthate per liter. This pH value, 9.8, is identical with the critical pH value for galena if lead salts are absent from solution. Moreover, sphalerite in the same conditions has its critical pH value raised from 8.6 (absence of carbonate) to 9.8 (carbonate present). If a large excess of carbonate is added it will prevent activation of sphalerite by lead. Carbonate has no effect upon the critical pH value of galena or of lead-preactivated sphalerite in the absence of the lead salt.

The behavior of minerals activated by lead is simpler than that of the same minerals activated by other cations because lead and lead-activated minerals are not depressed by cyanide. The activation of a first group of minerals—viz., sphalerite, chalcopyrite, pyrite—is of a similar nature. Sphalerite is activated and/or preactivated by solutions containing 199 mg. $Pb(NO_3)_2$ per liter at pH values from 2 to 13 but is not activated in very acid solutions ($pH < 2$). This tendency toward nonactivation in acid solutions is shown in a more marked fashion by pyrite; it is only possible to activate or preactivate this mineral by lead salts at pH values greater than 7.0. The response of pyrite is not connected with the beginning of activation because tests with higher concentrations of xanthate showed that it is possible to raise the critical pH value as high as one desires, while the critical pH value for preactivation is independent of xanthate concentration.

If the pyrite is preactivated, washed and then tested in a lead-free solution containing 25 mg. potassium ethyl xanthate per liter, a critical pH value of 9.0 is found. If sphalerite, chalcopyrite, and galena are treated by this method, critical pH values of 9.8, 11.0 and 9.8, respectively, are found, the response in each instance being independent of cyanide concentration. Hence preactivated sphalerite behaves identically with galena while preactivated chalcopyrite exhibits a "super-galena" and activated pyrite a "sub-galena" behavior. The minerals of a second group consisting of pyrrhotite and covellite (Fig. 5) behave differently. Tests with covellite yielded the usual type of cyanide-pH value curve; because of precipitation of most of the xanthate as the lead salt the tolerance to cyanide is lower than usual. The deduction from the type of curve is that lead salts are unable to activate covellite under the conditions chosen.

Pyrrhotite is a "cyanicide" in gold cyanidation; addition of a small amount of lead acetate or nitrate prevents consumption of cyanide. This suggests that the pyrrhotite acquires a lead-bearing surface film, but all the usual tests to distinguish between activated and unactivated pyrrhotite failed. The behavior of covellite indicates that copper sulphide is not activated by lead salts. This may help in considering chalcopyrite in which probably the iron of the crystal lattice, but not the copper, is replaced by lead. This may well lead to the "super-galena" behavior under preactivation conditions. The hypothesis that the copper and iron in the chalcopyrite surface react independently is further strengthened in that preactivation and/or activation is possible only in alkaline solutions, a behavior that is characteristic of pyrite.

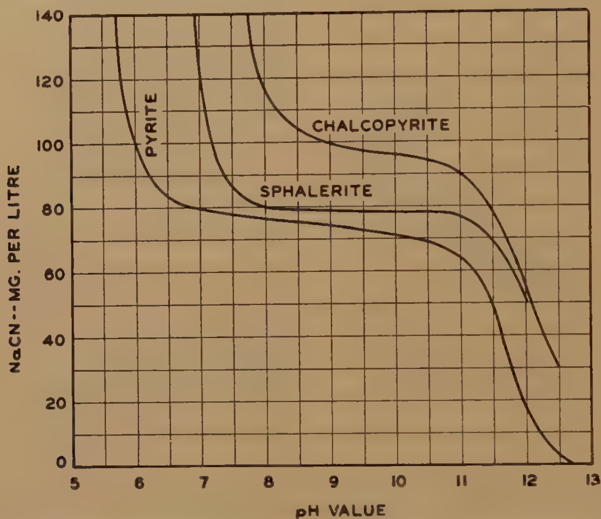

FIG. 6.—RELATIONSHIP BETWEEN CONCENTRATION OF SODIUM CYANIDE AND pH VALUE NECESSARY TO PREVENT CONTACT AT MINERAL SURFACES.
KEtX, 25 mg. per liter; temperature, 35°C.

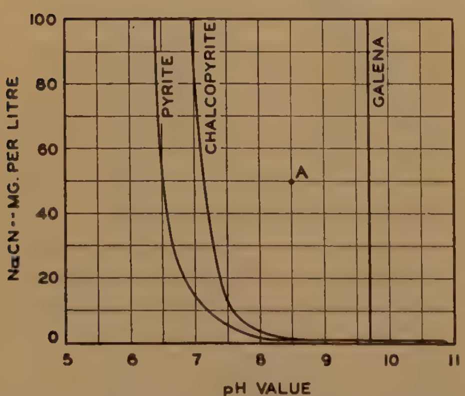

FIG. 7.—RELATIONSHIP BETWEEN CONCENTRATION OF SODIUM CYANIDE AND pH VALUE NECESSARY TO PREVENT CONTACT AT MINERAL SURFACES.
$CuSO_4.5H_2O$, 150 mg. per liter; KEtX, 25 mg. per liter; Na_2CO_3, nil; temperature, 35°C.

Activation and/or Preactivation of Sphalerite by Copper Sulphate and Lead Nitrate

If a solution of composition given by a point such as *A*, Fig. 6, is chosen a chalcopyrite or pyrite surface can be distinguished from a galena or galena-like surface by a contact test.

Sphalerite that has been preactivated by copper rather than lead will behave similarly to chalcopyrite or covellite, so that it will be possible to distinguish between galena-like and chalcopyrite (covellite)-like coatings on the mineral. Tests were made to ascertain the effect of lead salts on sphal-

erite preactivated by copper sulphate. After immersion in a solution containing men was again washed and placed in a xanthate solution of composition given by

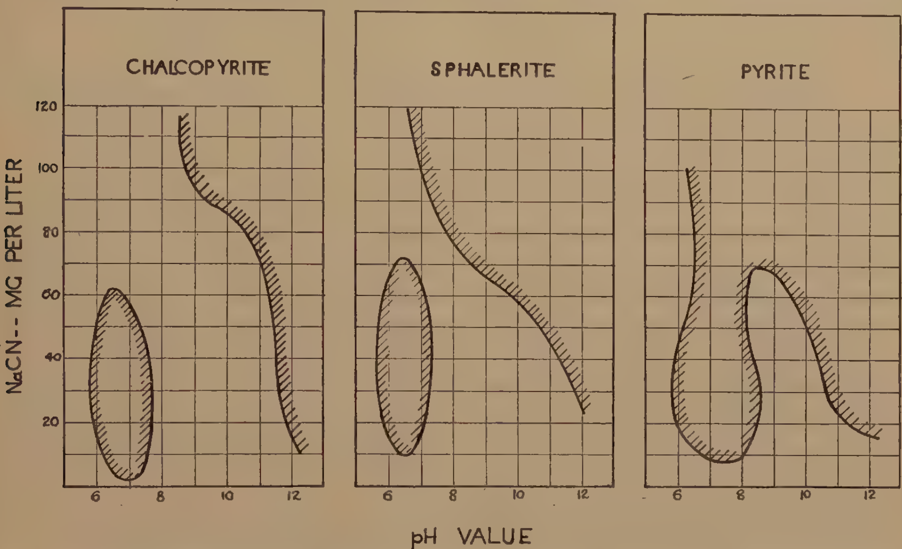

FIG. 8.—RELATIONSHIP BETWEEN CONCENTRATION OF SODIUM CYANIDE AND pH VALUE NECESSARY TO PREVENT CONTACT AT MINERAL SURFACES.

$CuSO_4.5H_2O$. 150 mg. per liter; KAmX, 6.31 mg. per liter; Na_2CO_3, 150 mg. per liter; temperature, 35°C.

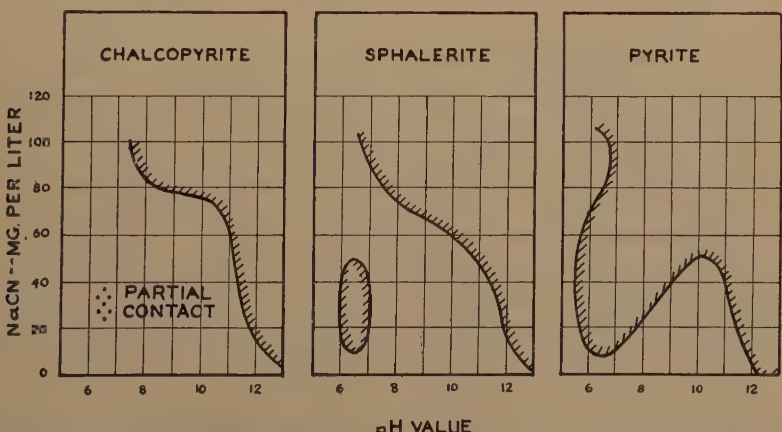

FIG. 9.—RELATIONSHIP BETWEEN CONCENTRATION OF SODIUM CYANIDE AND pH VALUE NECESSARY TO PREVENT CONTACT AT MINERAL SURFACES.

$CuSO_4.5H_2O$, 150 mg. per liter; KEtX, 5 mg. per liter; Na_2CO_3, 150 mg. per liter; temperature, 35°C.

150 mg. $CuSO_4.5H_2O$ per liter followed by washing, the specimen was placed in the required lead solution containing 199 mg. $Pb(NO_3)_2$ per liter. After 10 min. the specimen was again washed and placed in a xanthate solution of composition given by point *A* (Fig. 6). Two results were obtained:

1. If the lead solution was acid—whether cyanide-bearing or cyanide-free—the lead did not displace the copper.

2. If the lead solution was alkaline—again whether cyanide-bearing or cyanide-free—the mineral now behaved in a galena-like fashion. The lead had replaced the copper or had been adsorbed over it.

For the acid-lead solution, it was shown that the mineral had remained activated by copper, since contact could be obtained by lowering the pH value of the test solution.

The order of preactivation was now reversed. Sphalerite was preactivated in a lead-bearing solution, washed and placed in the required copper-bearing solution, again washed, and then tested in the xanthate solution. In this case the copper displaced the lead in both acid and alkaline solutions unless the cyanide concentration was sufficiently high to convert the copper ions into complex ions and so remove them. Thus the replacement of adsorbed copper by lead or of adsorbed lead by copper on a sphalerite surface is reversible in alkaline solutions.

Copper Sulphate—Amyl Xanthate

Fig. 7 shows the standard types of curves for minerals in the presence of copper sulphate. Departures, and the conditions of departure from this type of curve, have been discussed in an earlier paper[4] with results summarized as follows:

1. If the xanthate concentration and/or temperature are high the curves are of the "standard type" (Fig. 7).

2. Under certain conditions an "island" or second region of noncontact forms (illustrated by sphalerite in Fig. 9 of this paper). The formation of this island is favored by (*a*) low xanthate concentration, (*b*) high concentration of copper salt, (*c*) low temperature, (*d*) certain anions associated with the copper.

3. This island is due essentially to xanthate deficiency and not to lack of activation.

The size of the island of noncontact varies considerably with the collector used.[2,3,4] A comparison of the curves in Figs. 8 and 9 shows that under similar conditions island formation is of greater extent for amyl xanthate than for ethyl xanthate; i.e., the precipitation of copper amyl xanthate is more important than the greater collecting power of amyl xanthate.

Silver Sulphate

If the effect of silver salts in place of copper salts is studied, the second region of depression appears again but in a different region. Fig. 10*a* shows the standard type of curve for silver salts and should be compared with Fig. 7 for copper salts. One function of the cyanide is to remove silver ions from solution and then excess cyanide ions will act as a depressant for the chalcopyrite. Silver ions are removed in accordance with the equations:

$$Ag^+ + CN^- = AgCN$$
$$AgCN + CN^- = [Ag(CN)_2]^-$$

Hence if 187 mg. per liter is the concentration of silver sulphate in the conditioning solution, 118 mg. NaCN per liter is required to convert the silver ions into the complex argenticyanide. Thereafter the amount of cyanide ion available for the depression of the mineral depends upon the pH value of the solution (p. 202 of ref. 1). The standard curve is approximately a line of constant cyanide-ion concentration.

As the xanthate concentration is lowered (compare copper sulphate solution) the second region of depression makes its appearance at a pH value between 8 and 9 in the absence of cyanide. The curves for chalcopyrite show the growth of the second region as the xanthate concentration is lowered. The formation of this region of noncontact can be explained in the following manner. In the absence of cyanide virtually all of the xanthate is precipitated by silver ions. Hence only a low concentration of hydroxyl ions is required to prevent adsorption of the remaining soluble xanthate. As the concentration of hydroxyl ions is further increased, the silver is precipi-

tated as oxide and the xanthate-ion concentration increases. Hence contact may become possible again at high pH values (Fig. 11*b*). Addition of cyanide precipitates silver ions

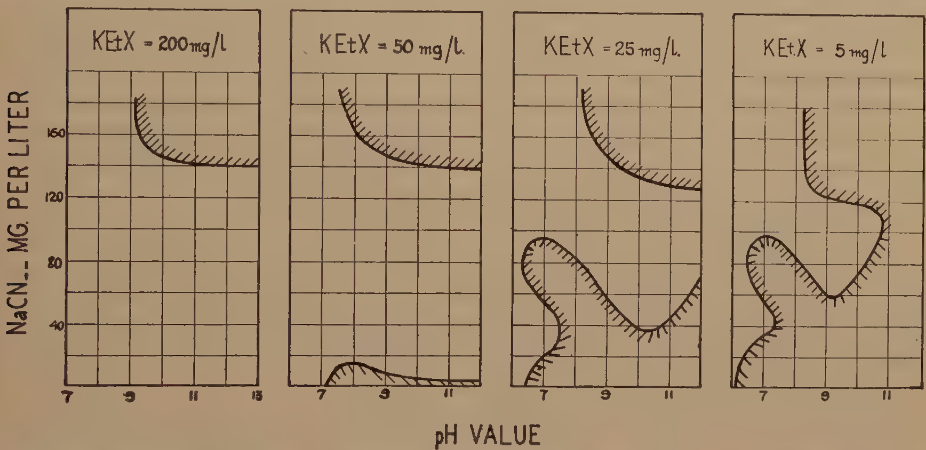

FIG. 10.—INFLUENCE OF ETHYL XANTHATE CONCENTRATION UPON CONTACT AT A CHALCOPYRITE SURFACE.
Ag_2SO_4, 187 mg. per liter; Na_2CO_3, nil; temperature, 35°C.

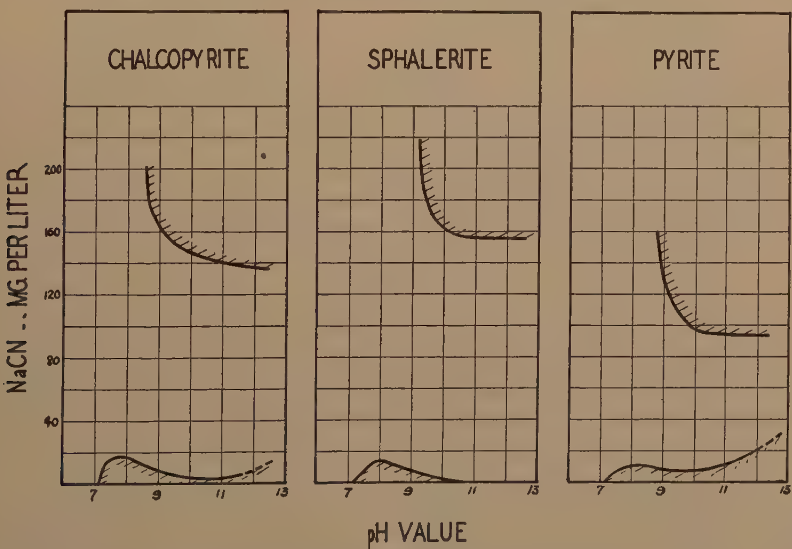

FIG. 11.—RELATIONSHIP BETWEEN CONCENTRATION OF SODIUM CYANIDE AND pH VALUE NECESSARY TO PREVENT CONTACT AT MINERAL SURFACES.
Ag_2SO_4, 187 mg. per liter; KEtX, 50 mg. per liter; Na_2CO_3, nil; temperature, 35°C.

and decomposes silver xanthate so that the xanthate-ion concentration rises. When the xanthate-ion concentration reaches a critical value, adsorption occurs and flotation is possible over a wide pH range. At still higher concentrations of cyanide, the cyanide ion acts as a depressant in the usual manner, so that the standard curve is obtained. If xanthate is in excess (192 mg. KEtX per liter is equivalent to 187 mg. Ag_2SO_4 per liter) the island region disappears.

If the curve for 5 mg. KEtX per liter is compared with the corresponding curve

when copper sulphate is present (Fig. 4*A* of ref. 4), it is noted that the second region of noncontact formation occurs to a far greater extent. Hence, in terms of the xanthate deficiency theory, the effective factor is a greater insolubility of silver ethyl xanthate, so that a more effective removal of xanthate ion occurs than when copper salts are used. Warren[6] has measured the precipitating powers of these salts and his results indicate that copper xanthate is approximately three times as soluble as silver xanthate.

The concentration of 50 mg. KEtX per liter was chosen as a satisfactory xanthate concentration to show the formation of the second region of noncontact for the other minerals studied, since chalcopyrite usually is the least susceptible to the formation of a second region of noncontact; i.e., if a small second region shows for chalcopyrite, larger regions usually will appear for pyrite and sphalerite. Fig. 11 indicates that the size of the second regions of noncontact for the three minerals is approximately the same but that the upper standard portions of the curves differ very considerably. The order of response is sphalerite (the most floatable) chalcopyrite and pyrite. This differs from the order in the absence of metallic ions or in the presence of copper sulphate.

Summary

The effects of cyanide and alkali upon adsorption of potassium ethyl xanthate at activated mineral surfaces has been investigated. It has been shown that:

1. Zinc sulphate has but a small influence upon adsorption in the presence of cyanide and has a considerable depressant action on chalcopyrite in the absence of cyanide.

2. For copper sulphate, the relative insolubility of copper amyl xanthate compared with copper ethyl xanthate induces greater island formation (second region of nonadsorption).

3. The formation of the second region of noncontact (nonadsorption) is more marked with silver salts, which is in accordance with the greater insolubility of silver ethyl xanthate compared with copper ethyl xanthate.

4. The order of response of various minerals to xanthate differs according to the cation present. For solutions containing cupric ion it is: chalcopyrite (most readily floated), sphalerite, pyrite; for solutions containing silver ion it is: sphalerite, chalcopyrite, pyrite.

5. Activation by lead salts produces galena-like surfaces; i.e., response to the collector is independent of cyanide. The character of the activation is dependent upon the mineral.

6. Covellite is not activated by lead salts.

7. Consecutive treatment of sphalerite with solutions containing lead ions and copper ions is studied. The changes are reversible in alkaline solution.

Acknowledgments

The writer gratefully acknowledges his indebtedness to the companies for which the work was carried out—viz.: Broken Hill South Ltd., North Broken Hill Ltd., Zinc Corporation Ltd., Electrolytic Zinc Co. of Australasia Ltd., Mt. Lyell Mining and Railway Co. Ltd., and the Burma Corporation Ltd.—and also to Dr. Ian W. Wark, for considerable help and criticism; to Mr. H. Hey, under whose general direction the work was carried out; and to Prof. E. J. Hartung, for providing laboratory accommodation.

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DISCUSSION

(*A. F. Taggart presiding*)

E. C. Peterson,* Salt Lake City, Utah.—An observation made a good many years ago may be worth recounting here as it probably confirms Mr. Sutherland's second conclusion, concerning the nonadsorption of amyl xanthate in the presence of copper sulphate. He says that "the precipitation of copper amyl xanthate is more important than the greater collecting power of amyl xanthate"; or, we may conclude that under certain similar conditions ethyl xanthate may have relatively greater collecting power than amyl xanthate.

In the selective flotation of a pyritic lead-silver ore, in which silver was associated with both galena and pyrite, a lead concentrate was obtained by depressing the pyrite with cyanide in a circuit maintained near neutrality (pH 7.0 to 7.5) by the addition of lime during grinding. The pyrite was reactivated for subsequent flotation by the addition of copper sulphate. Potassium ethyl xanthate was the principal soluble collector. A froth heavily loaded with pyrite was obtained when K-ethyl xanthate was used but no promotion of the pyrite was obtained and the froth was substantially barren and watery when K-amyl xanthate was substituted equally for K-ethyl xanthate under exactly parallel test conditions.

Although amyl xanthate is considered, and generally is found to be a stronger collector than ethyl xanthate, perhaps many can recall certain flotation circuits or certain conditions in actual mill practice in which ethyl xanthate alone has been found to possess greater collecting power than amyl xanthate used alone or in combination with ethyl xanthate.

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The Mechanism of Activation in Flotation

By A. M. Gaudin,* Member, and Alfonso Rizo-Patrón,† Student Associate A.I.M.E.

(New York Meeting, February 1942)

Previous studies of activation in flotation have directed attention to the action of the activator on the mineral to be floated rather than to the relationship of the activator to the collector. The latter phase of the problem, however, seems to have definite scientific, if not also practical, interest.

In the investigation reported in this paper, experiments were carried out, largely with quartz as subject, barium as activator, and oleate as collector. The results obtained indicate that a 1:1 ratio of collector ion to activator ion gives optimum results, and that a 2:1 ratio almost fails to achieve activation and flotation. Yet, a 2:1 ratio would have been expected on stoichiometric grounds, had the coating been a normal salt of the activator ion with the collector ion, since barium is a divalent cation and oleate a monovalent anion.

To interpret and rationalize the observations, considerations of the crystal chemistry of quartz are helpful. The point of view to which these considerations lead may be of general applicability to systems involving suspensions of solids in liquids; they confirm that an adsorbed coating is inescapable.

Apparatus and Supplies

The experimental work consisted in barium-ion abstraction tests, hydrogen-ion and hydroxyl-ion abstraction tests, flotation tests, and determinations of the surface of the crushed quartz.

The flotation machine was built according to the design of D. W. McGlashan.* It consists of a drill press in which the bit has been replaced by a stainless-steel impellor surrounding a stainless-steel stator, as in the Fagergren laboratory machine. The air intake is controlled by a valve, and the cell consists of a beaker.

The quartz was obtained from vein quartz stored in large chunks. It was first crushed to 20 mesh, then sized on a 28-mesh screen. The 20 to 28-mesh size alone was retained for use. This size was freed of coarse iron by passage through a Ball-Norton magnetic separator, bleached with boiling hydrochloric acid to remove residual abraded iron, washed with distilled water until the washings were thoroughly free of ferrous, ferric, and excess hydrogen ion, and stored in covered Pyrex beakers under water until used.

The barium was used as barium chloride, Baker's analyzed C. P.

The oleate was used as sodium oleate. It was prepared from sodium hydroxide, Baker's analyzed C. P., and oleic acid purified by R. R. de Arellano[3] in the laboratories at the Massachusetts Institute of Technology. The purification procedure was elaborated from the recent researches of Brown and Shinowara.[2] It consisted in: (1) saponification of olive oil with potassium

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Abstract from Bachelor's and Master's theses in Mineral Dressing Laboratories at Massachusetts Institute of Technology by A. Rizo-Patrón, 1940 and 1941, respectively. Manuscript received at the office of the Institute Nov. 19, 1941. Issued in Mining Technology, May 1942.

* Richards Professor of Mineral Dressing, Massachusetts Institute of Technology, Cambridge, Mass.

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[2] References are at the end of the paper.

hydroxide, (2) neutralization of the soap with hydrochloric acid, (3) esterification of the crude fatty acids with methanol, (4) low-pressure fractional distillation (15 mm. Hg) of the crude ester to yield pure methyl oleate, (5) double fractional crystallization of methyl oleate from acetone solution at minus 30°C. and minus 55°C., (6) storage of the purified methyl oleate in sealed glass ampoules in a refrigerator to prevent oxidation and polymerization, (7) saponification of the methyl oleate, distillation of the resulting methanol, acidification and exact neutralization just prior to use.

Abstraction of Barium Ion by Quartz

Two series of tests were made to establish that quartz abstracts barium ion from aqueous solution.

In the first series, the quartz (about 50 grams) was ground in a pebble mill with porcelain pebbles for a standard grinding period (25 min.) and transferred to a beaker; barium chloride solution was added after adjustment of the pH. Following a reaction period, the pulps were filtered and washed with wash solution at the same pH as the reacting pulp. The barium ion withdrawn from solution crudely approximated the quantity adsorbed. We say "crudely" because the experiments were limited by: (1) possible retention of interstitial dissolved Ba^{++} in spite of washing, (2) possible shift in adsorption equilibrium during washing, (3) possible removal of Ba^{++} as carbonate by atmospheric carbon dioxide, where the circuit was alkaline. The results obtained are presented in Table 1. These results show that barium-ion abstraction is a function of pH, being nil in acid circuit, smaller in nearly neutral circuit but substantial in alkaline circuit. They show, also, that the abstraction depends upon the extent of surface of the mineral.

In the second series of tests no washing of the filter cake was performed, barium-ion abstraction being corrected for the weight of filtrate retained by the filter cake. The results obtained are presented in Table 2. These experiments establish that Ba^{++} in alkaline circuit is adsorbed by quartz and give a measure of the extent of this adsorption.

TABLE 1.—*Abstraction of Barium Ion by Quartz*
FIRST SERIES

Determination	Ba^{++} Added (Expressed as Mg. of $BaCl_2$)	Quartz Added, Grams	pH during Adsorption[c]	Ba^{++} in Filtrate (as Mg. $BaCl_2$)	Ba^{++} Adsorbed	
					As Mg. $BaCl_2$	Micromol per Gram of Quartz
a	100.0	50[a]	10.75	81.9	16.8	1.61
b	100.0	58[b]	10.75	91.6	7.1	0.59
c	100.0	59[b]	5.70	97.4	1.8	0.146
d	100.0	55[b]	0.0	99.2	0.0	0.0
e	100.0	0	10.75	98.7	0[d]	

[a] Not deslimed; ground for the standard period of 25 minutes.
[b] Deslimed by siphoning off slimes five times, the settling time corresponding to a particle size of about 12 microns, after grinding for the standard period of 25 minutes.
[c] Determined with a Beckman pH meter.
[d] Assumed.

Abstraction of Hydrogen and Hydroxyl Ions by Quartz

Addition of acid or alkali to suspensions of quartz failed to give a pH as far removed from neutrality as was expected. When acid is added to quartz, the pH obtained is that which would result if there were abstraction of hydrogen ions and/or emission of hydroxyl ions by the quartz. Conversely, when an alkali is added, the result can be interpreted as abstraction of hydroxyl ions and/or emission of hydrogen ions. Fig. 1 shows the results obtained in a pair of experiments.

Since the abstraction or emission of one ion without compensating abstraction or emission of some ion of opposite charge is electrostatically inacceptable, the authors change in the effect reported above. Since it has been shown that barium ion is not abstracted by quartz from an acid circuit, this accords with the postulate.

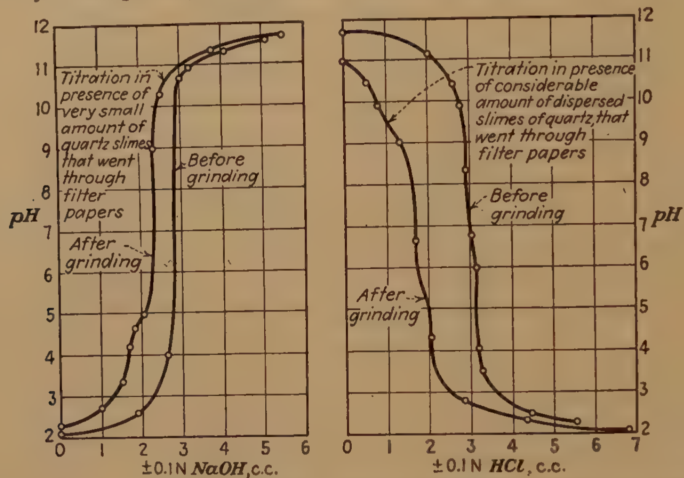

FIG. 1.—TITRATION OF EQUAL VOLUMES OF FILTRATES FROM PULPS OF QUARTZ IN WATER, AFTER ADDITION OF A DEFINITE QUANTITY OF ACID (LEFT) OR OF ALKALI (RIGHT), BEFORE AND AFTER GRINDING.

In both cases the increased surface due to grinding has resulted in a decrease in titer at equivalent pH. All volumes used in titration were 25 cubic centimeters.

propose: (1) that quartz in water is covered with hydrogen *and* hydroxyl ions, this cover being substantially complete; (2) that when hydrochloric acid is added, chloride ion takes the place of hydroxyl ion; (3) that when sodium hydroxide is added, sodium ion takes the place of hydrogen ion.

Barium ion added as chloride, in conjunction with hydrochloric acid leads to no

Barium ion added as chloride in conjunction with sodium hydroxide, again, fails to alter, qualitatively at least, the pH effect described. But since barium ion is abstracted by the quartz, if the exchange is regarded as occurring between barium and sodium ions it is clear that the addition of barium chloride should fail to change the pH drift caused by the mineral.

TABLE 2.—*Abstraction of Barium Ion by Quartz*

SECOND SERIES

Determination	Ba^{++} Added (Expressed as Mg. of $BaCl_2$)	Quartz Added,[a] Grams	pH during Adsorption	Ba^{++} Remaining in Solution[b] (as Mg. $BaCl_2$)	Ba^{++} Adsorbed	
					As Mg. of $BaCl_2$[c]	Micromol per Gram of Quartz
	100.0	48.04	10.75	79.2	12.7	1.27
g	100.0	47.43	10.75	79.4	12.5	1.27
h	100.0	0.0	10.75	91.9	0[d]	
i	100.0	0.0	0.0	91.7	0[d]	

[a] The quartz was obtained by splitting a ground, partly deslimed pulp, a fraction of which was used for making a surface determination (see text).
[b] After allowing for retention of filtrate by the mineral cake.
[c] This allows for the Ba^{++} adsorbed by the filter paper, and for possible conversion to $BaCO_3$ by atmospheric CO_2.
[d] Postulated.

Quartz Surface Available for Barium Adsorption

One of the fractions of the deslimed quartz (used also in determinations *f* and *g*, Table 2) was sized by screening to 400 mesh and by the pipette method for fine sizes.[11] Because the pulp had been carefully deslimed, the weight of quartz finer than 10 microns was very small; as a result, the authors believe that the surface determination of spheres of equivalent mass calculated in Table 3 is more accurate than determinations on slimy pulps. They estimate that this surface may be accurate to ±5 per cent. This surface, however, neglects the irregularities in shape of broken fragments. According to the data of Gross and Zimmerley,[6] the actual surface of sieve fractions is larger than calculated by a variable factor. This factor was found by them (Table 1 in ref. 6) to decrease gradually as particle size was reduced from over 8 at 3 mesh to 3.11 at 28 to 35 mesh, 2.15 at 150 to 200 mesh, and 2.02 at 200 to 270 mesh. Clearly, the factor cannot decrease at extreme fineness below some theoretical minimum. It seems reasonable to consider this theoretical minimum as that provided by a block shaped in the fashion of a unit cell, whose edges are all of the same length. For such a block the calculated minimum is 1.31. Since the shape factor must vary with size, gradually rather than suddenly, assumed values of the shape factor are inserted in the fourth column of Table 3. The values are those of Gross and Zimmerley down to the 200 to 270-mesh grade (−74 + 52 microns) and values extrapolated from an enlarged graph of their values for finer sizes.

Table 3 shows that the true surface of the quartz must have been near 1240 sq. cm. per gram. As there was no evidence that the quartz used in these tests had the same shape distribution as Gross and Zimmerley's quartz, and as the bulk of the material was fractionated by sedimentation rather than by screening, the present estimate of surface may be in error by well over 10 per cent. Accordingly, the value of the surface will be retained to only two significant places and taken as 1.2×10^3 sq. cm. per gram.

Table 3.—*Sizing Analysis and Specific Surface of Ground Quartz Used in Making Barium-abstraction Determinations f, g*

Size, Microns	Weight, Per Cent	Surface of Spheres of Equivalent Size, Sq. Cm. per Gram	Shape Factor	Estimated Actual Surface Provided, Sq. Cm. per Gram of Sample
−147[a] + 104	0.23	0.5	2.37	1.2
−104 + 74	5.14	13.0	2.15	28.0
−74 + 52	11.43	41.1	2.02	83.0
−52 + 37	32.30	164.3	1.93	317.0
−37[a] + 27.6[b]	32.79	229.7	1.85	425.1
−27.6 + 19.4	14.92	143.8	1.77	254.7
−19.4 + 13.6	2.64	36.2	1.70	61.5
−13.6 + 9.5	0.13	2.5	1.64	4.1
−9.5 + 3.6[b]	0.28	9.5	1.56	14.8
−3.6[c]	0.14	31.6	1.50	47.4
	100.00			1,240.8

[a] Determined by screening.
[b] Determined by sedimentation, using the pipette method.
[c] Average size assumed to be 1.0 micron.

Flotation Experiments

Experiment shows that for satisfactory recovery the pH must exceed 10. This is in agreement with the abstraction experiments. Results are presented in Table 4.

Table 4.—*Effect of* pH *on Flotation of Quartz*

Barium Chloride, Lb. per Ton	Sodium Oleate, Lb. per Ton	Nujol, Lb. per Ton	pH of Original Cell	pH of Tailing Liquor	Quartz Recovered, Per Cent
1.0	1.5	1.36	7.2	6.9	0
1.0	2.0	1.36	9.7		47
1.0	2.0	1.36	10.5	10.0	83
1.0	2.0	1.36	11.5	11.1	91

At a pH in the suitable range (say 10 to 11) good flotation results are obtained if the ratio of mols of collector to mols of activa-

tor is kept in the vicinity of 1:1, but the results are poor in the region of stoichiometric equivalence. This is shown by Fig. 2, which records early results obtained with

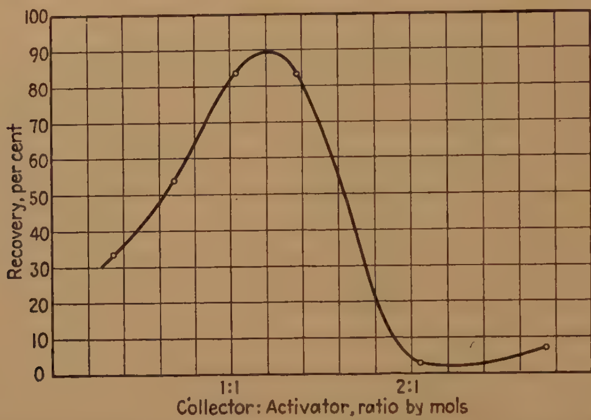

Fig. 2.—Flotation of quartz activated by barium ion with crude oleic acid as collector in terms of molar ratio of collector to activator.

pH, 10.6. Activator, $BaCl_2$, 1.0 lb. per ton. Nujol, 1 drop per pound soap per ton. Time of grinding, 30 min. No Terpineol used.

A celluloid flotation cell of the agitation type was used in these tests.

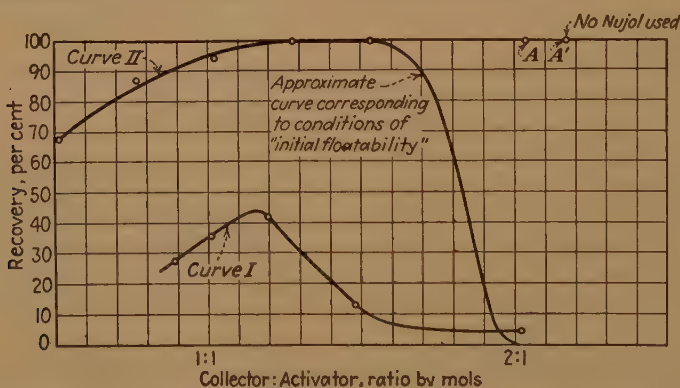

Fig. 3.—Flotation of quartz activated by barium ion with pure sodium oleate as collector in terms of molar ratio of collector to activator.

Curve I. Surface of quartz ±75 per cent saturated with barium. pH, 11.4. Terpineol used, 0.5 lb. per ton. No Nujol used. Time of grinding, 25 min. Quartz not deslimed.

Curve II. Surface of quartz ±100 per cent saturated with barium, pH, 10.75. Terpineol used, 0.15 lb. per ton. Nujol used, 2.06 lb. per ton. Time of grinding, 45 min. Quartz deslimed.

A and A′, recoveries obtained only when excess of soap had been removed, breaking up the weak adhesion between the first and second "coatings" of oleate ions on the surface of the quartz. This amounts to having destroyed the initial conditions of the respective tests by altering the ratio Collector: Activator present at the beginning of these tests.

crude oleic acid and by Fig. 3, which presents the results obtained with pure sodium oleate.

With reference to curve II, Fig. 3, attention is called to the fact that good recoveries with a large quantity of collector were obtained if the excess collector was allowed to overflow as a result of the excessive frothing in the system. At the beginning of floating the froth was watery and bare, but it became mineralized after the excess soap had overflowed.

Discussion of Results

The experimental results of these tests indicate:

1. In alkaline pulp quartz emits hydrogen ion and/or adsorbs hydroxyl ion.
2. In alkaline pulp, in the presence of barium ion, quartz adsorbs barium ion.
3. Effective flotation is obtained in alkaline pulp only.
4. Effective flotation with oleate requires an oleate: barium ratio of 1:1.

In addition data were obtained as to the surface of the quartz that is available for barium adsorption, and the crystal structure of quartz is well known.

The crystal structure of quartz is best understood in terms of the new ideas of crystal chemistry presented by Evans[4] or Stillwell.[12] These authors, as well as others, classify crystals into several classes: (1) molecular crystals in which the building stones are molecules, (2) ionic crystals in which the building stones are ions, (3) valence crystals, in which the building stones are atoms, and (4) metals, in which the building stones are atoms partly ionized to cations and electrons.

At first, quartz was believed by some investigators[8] to consist of molecules of SiO_2 arranged in suitable symmetry, and by others[7] to consist of atoms of silicon and of oxygen covalently bonded to form an endless, giant molecule.

The actual arrangement of the silicon and oxygen was worked out in detail by Bragg and Gibbs[1] and by Gibbs.[5] Fig. 4, showing a quartz crystal in three projections, is from the work of Gibbs. Each silicon is surrounded by four oxygens, and each oxygen is connected to two silicons. Such an arrangement is possible if quartz is made of atoms of oxygen and of atoms of silicon—in which case it is a valence crystal—or of Si^{4+} and O^{2-} ions—in which case it is an ionic crystal.

In recent studies[13] it has been pointed out that since the bonds emanating from the oxygens are 144° instead of 180° apart (ionic structure) or 90° apart (covalent structure), the structure must be of intermediate character. This is in line with the new ideas expressed by Pauling[10] and described by the term "resonance."

Fig. 4 shows that the structure repeats itself for each quadrangle $ABCD$, $A'B'C'D'$, and $A''B''C''D''$. Each quadrangle contains on the topmost layer one quarter of each of four silicons; that is, the equivalent of one silicon. Four bonds emanate from each silicon, two into the paper and two out of it. Thus, rupturing a quartz crystal in each of the three orientations gives two foci of unsaturation for each unit area. These unit areas are equivalent to $ABCD$, $A'B'C'D'$. and $A''B''C''D''$, respectively, for sections perpendicular to the c axis, perpendicular to one of the a axes, and parallel to an ac plane.

$ABCD$ is a lozenge of apical angle 60° and edge of 4.903 Å.; $A'B'C'D'$ is a rectangle with sides of 4.903 and 5.393 Å.; and $A''B''C''D''$ is a rectangle with sides 4.25 Å.; and 5.393 Å. These various unit quadrangles have areas of 26.4, 22.9, and 20.9 Å^2. The average area is 23.4 Å^2., if the unit quadrangles are equally abundant.

In other words, the loci of ruptured bonds occur with a frequency of two loci on each and every 23.4 Å^2. of surface. Since one locus has anionic preference (next to a silicon) and another cationic preference (next to an oxygen) there is room for chemosorption at the rate of one ion of each sign for every 23.4 Å^2. of surface. In water, for instance, $\frac{10^{16}}{23.4}$, or 4.27×10^{14} hydrogen ions and 4.27×10^{14} hydroxyl ions can be accommodated simultaneously per square centimeter of surface.

Now that an accurate estimate of the number of loci for chemosorption (two, one of each sign, for every 23.4 Å^2. of surface) has been obtained it is pertinent to inquire what may happen to quartz, broken under water. Such a surface, as may be seen

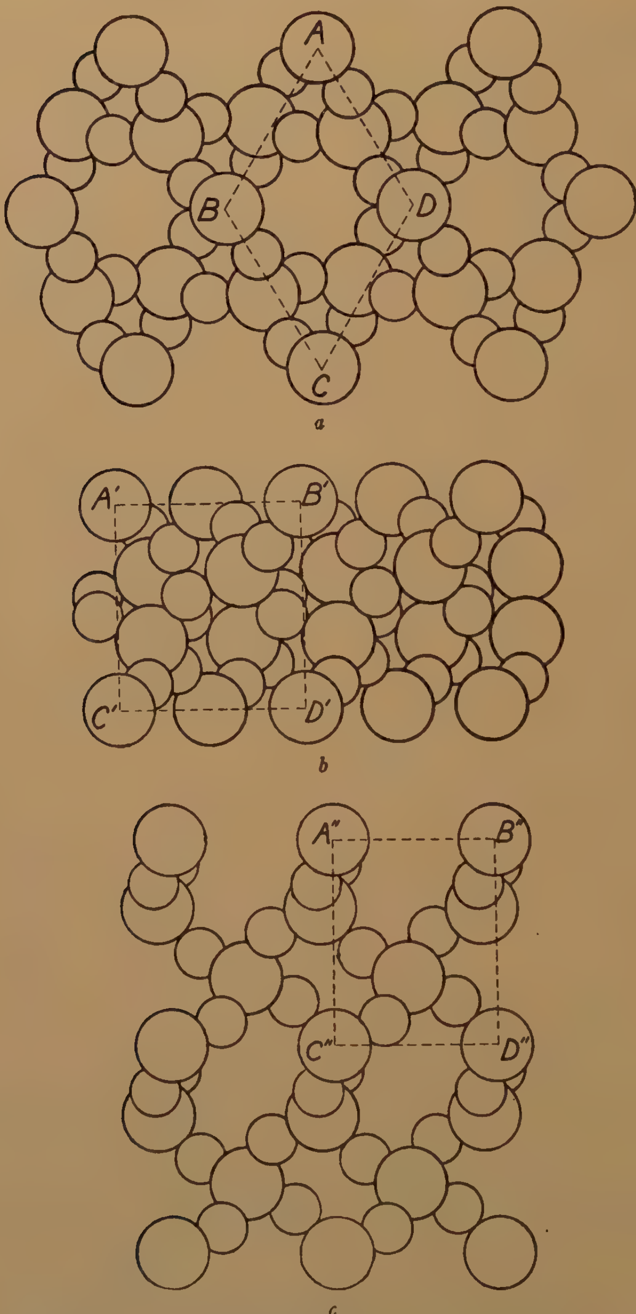

FIG. 4.—CRYSTAL STRUCTURE OF ORDINARY (ALPHA) QUARTZ.
Drawn to scale of 50,000,000 : 1, partly after Gibbs.
a, along *c* axis; *b*, perpendicular to *a* axis; *c*, parallel to an *a* axis.

from Fig. 5, can accommodate an H^+ ion opposite each oxygen and an OH^- ion opposite each silicon. The heat of wetting of quartz is evidence that this chemosorption is a strongly exothermic reaction. A quartz surface coated with hydrogen and hydroxyl ions, furthermore, is statistically neutral, the number of anions adsorbed balancing exactly the number of cations adsorbed. The resonance principle of Pauling[10] suggests that the oxygens resonate between successive silicons in such a way as to be closer to one silicon for a short time, then closer to the other. The same picture can be extended to the OH layer formed by the adsorbed H^+ and OH^-, thus suggesting the identity of all OH groups, over a given time interval, each group behaving now cationically, then anionically.

Thus one may postulate that the maximum barium-ion adsorption of which quartz should be capable is that indicated by attachment of one barium ion at every locus of ruptured bond on the quartz surface.

This assumption makes the maximum adsorbability of barium ion

$$2\frac{(1.2 \times 10^3)(4.27 \times 10^{14})}{6.06 \times 10^{23}} = 1.7 \times 10^{-6} \text{ mol}$$

or 1.7 micromol.

The experimental figure of 1.27 (Table 2) for conditions not necessarily maximal should be considered as in good accord.

In particular, it should be noted that if a barium ion were to replace stoichiometrically two H^+, instead of replacing the H from every H^+ *and* every OH^-, the calculated maximum adsorbability would be 0.42 micromol, or approximately one third of the observed value.

The oleate involved is generally regarded as having been abstracted from solution by the mineral. If the coating had been formed by a chemical reaction of well recognized type—viz., by methathesis—it would have consisted of $Ba(Ol)_2$ and the oleate ions required would be twice the barium ions, not equal to them. As a matter of fact, it is difficult to see how $Ba(Ol)_2$ would "hang on" to quartz if oleate ion alone does not do so, since all the valencies

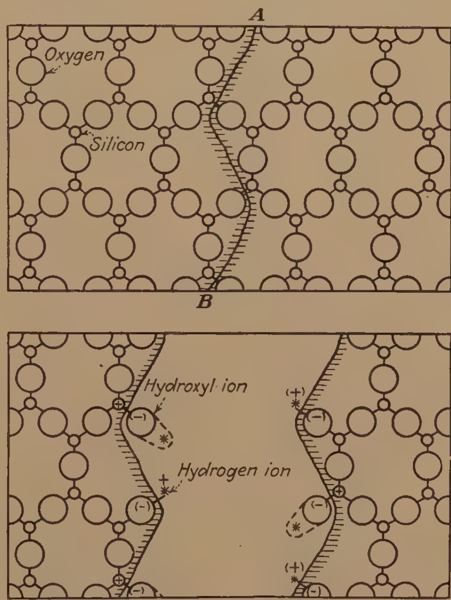

FIG. 5.—SCHEMATIC REPRESENTATION OF SILICON-OXYGEN BONDING IN QUARTZ AND THE ADSORPTION OF H^+ AND OH^- IONS THAT RESULTS WHEN IT IS BROKEN UNDER WATER.

As the connections between oxygens and silicons are shown in a plane rather than three-dimensionally, only three of the four valences of the silicons are represented.

—the tentacles—of the barium would be occupied.

But if the Ba:Ol ratio is 1:1, one valency of the barium remains available for attachment and thus provides a mechanism for fastening oleate ions to quartz. This, of course, is equivalent to postulating adsorption of the monovalent cation $(BaOl)^+$ in place of the sequential adsorption of the divalent cation Ba^{++} and the monovalent anion Ol^-. Although it cannot be said that that adsorption of $(BaOl)^+$ does not take place, the sequential adsorption appears preferable, since Ba^{++} is adsorbed in the absence of Ol^-. Perhaps in that case the adsorbed ion is $(BaOH)^+$.

In any case, the net effect is equivalent to flotation by a coating of $(BaOl)^+$. It is interesting to observe that flotation of quartz by $(BaOl)^+$ is flotation by a cationic collector, just as quartz flotation by laurylammonium ion $(NH_4La)^+$ is acknowledged to be cationic-collector flotation.

It is also to be noted that if the phenomena described in this paper are general for activation operations, they offer the possibility that nonactivated minerals are actually floated by a collecting cation formed from the anion (monovalent) of the reagent and the cation (divalent) of the mineral. For example, galena might be regarded as floated by $(PbX)^+$ rather than by X^- when xanthate is the collector added. Actually, preliminary experiments with sphalerite, cupric ion and ethyl xanthate ion verify this expectation.

Acknowledgment

The writers wish to acknowledge the opportunity they had of discussing this subject with Dr. Martin J. Buerger, Professor of Mineralogy, and Dr. R. Schuhmann, Jr., Assistant Professor of Mineral Dressing, at the Massachusetts Institute of Technology. Their thanks go to both.

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DISCUSSION

(H. R. Hanley presiding)

N. Arbiter,* New York, N. Y.—The inference is drawn from the reported flotation tests that because a maximum recovery was found at about a 1:1 molar ratio of oleate-barium ion added to the pulp with poorer recoveries at ratios less and greater than this value, it follows that the ratio of oleate to barium ions at the mineral surface may have this same value. Actually, as far as conditioning is concerned, the data show that the mineral became completely floatable at about a 1:1 ratio and remained completely floatable to beyond a 2:1 ratio. The authors admit this in attributing poor recovery at the 2:1 ratio not to lack of filming by a primary soap layer but to depression by excess. Hence there is no difference in conditioning between 1:1 and 2:1 ratios of oleate to barium.

The question may be raised as to the significance of the lower recoveries where the oleate to barium ratios were less than 1:1. The authors added barium chloride solution to the quartz pulp followed by sodium oleate solution in the required amount. (Information supplied through the courtesy of Professor Gaudin.) Under these circumstances it is to be expected that a precipitate of barium oleate will form, since the amount of barium ion added is far greater than that needed to activate the quartz. In the cases where the barium ion concentration is in stoichiometric excess of oleate (oleate-barium ratio less than 2:1), the concentration of oleate in solution will be substantially reduced according to the mass action law. The original ratio of mols added cannot be maintained since for each molecule of barium precipitated twice as many of oleate are removed. Poor recovery in these cases may be due to low concentration of dissolved oleate.

The authors propose as mechanisms for the activation and flotation reactions either that a $Ba(OH)^+$ complex activates the quartz, replacing an H^+ or a Na^+, followed by an oleate

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ion replacing the $(OH)^-$ or that a $Ba(Oleate)^+$ complex directly replaces an H^+ or a Na^+. They argue that collection by metathesis, which they interpret as due to $Ba(Oleate)_2$ is opposed by the poor recoveries at the 2:1 oleate to barium ratio while collection by the complex with the 1:1 oleate to barium ratio is supported by the good recoveries at the 1:1 ratios of oleate to barium. It has already been suggested that the poor recoveries at the 2:1 ratio are not related to the collection reaction and that at the 1:1 ratios of added reactants where good recoveries were obtained the actual ratio of oleate to barium in solution was probably different from 1:1 due to precipitation of $Ba(Oleate)_2$. There is thus little support for the mechanism advanced.

Apart from this, collection by metathesis does not imply the formation of $Ba(Oleate)_2$, which then attaches itself to the mineral, but rather the replacement of an inorganic surface ion by a collector ion in stoichiometric proportion. This is precisely what the authors postulate for the behavior of their complex cations. It would seem, therefore, that in so far as the particular phenomena under discussion are concerned there is no distinction between the terms exchange adsorption and metathesis.

G. Gutzeit,* Westport, Conn.—This paper has a high scientific standard and is a very interesting and valuable contribution toward the understanding of one of the complex phenomena resulting in flotation. It proposes an ingenious and attractive theory in order to explain the mechanism of activation.

The conclusions are based on the fact that quartz abstracts barium ions in alkaline pulp, and can then be floated, using sodium oleate as a collector, the optimum ratio of mols of sodium oleate to mols of barium being 1:1 instead of one barium to two oleates, as would be expected if the flotation were due to the formation of normal barium oleate (following stoichiometric equivalence). The authors postulate that the binding of the oleate to the quartz surface is accomplished by means of the complex cation $(BaOl)^+$, and not by a coating of $Ba(Ol)_2$. They state that "it is difficult to see how $Ba(Ol)_2$ would 'hang on' to quartz if oleate ion alone does not do so, since all the valencies—the tentacles—of the barium would be occupied."

This theoretical reasoning is not quite convincing, $Ba(Ol)_2$ being a dipole and having an asymmetrical electric field which may permit a molecular binding. However, their experimental results do support their theory quite well. I would like to call attention to a paper by L. Kraeber and A. Boppel,[15] containing experimental data which, when properly interpreted, show that the ratio of mols of collector to mols of activator ($Fe^{\cdots}$) is lower than the value of the normal soap. This gives additional support to Gaudin's and Rizo-Patrón's hypothesis.

Kraeber and Boppel study the activation of quartz with many metallic salts, using Monopolsoap (sodium ricinol sulphate

$$CH_2{\cdot}(CH_2)_5{\cdot}CH{\cdot}(OSO_3Na){\cdot}CH_2{\cdot}CH{:}CH \quad (C_2H_2)_7{\cdot}COOH)$$

as the collector.

In some instances, the optimum amount of the activating metal-salt solution has been determined, when an arbitrarily chosen quantity of soap of 5.0 c.c. M/100 was used. Table 5 abstracts some figures reported by Kraeber and Boppel. (The pH values are taken from their curves.)

These data evidently were not intended to evaluate the optimum ratio of collector to activator for flotation, but at least they suggest that it would be worth while to repeat the accurate tests conducted by A. M. Gaudin and A. Rizo-Patrón, using other metallic salts as activators, in order to see whether the theory of these authors has a general meaning, or if it applies only to certain specific reagent combinations.

It is interesting to note that a critical analysis and recalculation of the experimental results of these German authors on the *adsorption* of ferric cations also leads to the conclusion that there is no stoichiometric equivalence between the optimum amount of the *adsorbed* activating ion and that of the collector; in other words, that the flotation is not due to the formation of a normal iron sulphoricinate. Their tests show that the abstraction of iron from a solution of $FeCl_3$ is of the order of magnitude of a monionic layer (Table 3, p. 423 of ref. 15). The ratio of mols of collector (sodium ricinol sulphate) to

* The Westport Mill Research Division, The Dorr Company.

[15] L. Kraeber and A. Boppel: *Metall und Erz* (Oct. 1934) Heft 19, 31.

mols of *adsorbed* iron is 1:2* (instead of 3:1, as would be expected if the flotation were due to the formation of normal iron sulphoricinoleate). In other words, only one molecule of the sulphated soap is used for every two ions of activating ferric ion.

It seems risky to try any chemical interpretation of this fact. The constitution of the Monopolsoap suggests that each cation could be attached to the quartz surface by two valencies, and that the fatty-acid sulphate could form a bridge, binding the residual valence of one iron by means of the carboxyl group, and the other one by the sulpho group. Nevertheless, it is doubtful whether, at a pH as low as 3.5, the carboxyl group would react at all.

Kraeber and Boppel used 15 grams of sized quartz (0.06 to 0.088 mm.) by for their flotation tests. If the amount of adsorbed iron is calculated as a function of the surface of the quartz, the result (0.39 mg.) checks fairly well the experimental values (0.37–0.42 mg.). From the minimum amount of Monopolsoap necessary to get good flotation results, and assuming that the sodium ricinol sulphate has the same molecular diameter as oleic acid (i.e., 46 Å^2), 1.13 mg. is needed in order to form a monomolecular layer. (The experimental value is 1.26 mg.) Thus, the calculated molar ratio of collector to activator without any other hypothesis than the formation of a monomolecular layer of the collector and a monoionic layer of the activator is also 1:2.

It can be stated that the experiments of Gaudin et al., as well as the tests of Kraeber and Boppel, prove that, for the flotation of quartz, using a cation as activator and a soap as collector, the molar amount of collector is definitely lower than the quantity that would be required for the formation of a normal metallic soap by metathesis. This fact shows once more that classical chemical reasoning does not always apply to flotation.

TABLE 5.—*Data from Kraeber and Boppel*

Activating Salt	Amount of M/100 Solution[a] of Soap, C.C.	Amount of M/100 Solution of Salt, C.C.	Optimum pH	Flotation
Copper chloride	5.0 (3.7)	7.0	6.0– 9.0	Good
Copper sulphate	5.0 (3.7)	7.0	6.0– 9.0	Good
Magnesium chloride	5.0 (3.7)	5.0	9.5–12.0	Good
Calcium chloride	2.7 (2.0)	7.0	9.5–14.0	Very good
Lead acetate	5.0 (3.7)	0.9 ($PbCl_2$)	5.7– 9.7	Very good
Ferric chloride	0.4 (0.296)	1.3	3.5– 7.5	Very good

[a] The molecular weight of sodium ricinol sulphate is 423. As Kraeber and Boppel used 3.14 grams of Monopolsoap per liter, they were really working with a 0.0074 M solution.

* 0.42 mg. of $Fe^{\cdots}$ adsorbed on 15 grams quartz (of 0.06 to 0.088 mm. size) and 1.26 mg. of soap used for the flotation: 7.55×10^{-6} mols adsorbed; viz., 3×10^{-6} mols soap used as collector.

Experiments with Slime-coatings in Flotation

By S. G. Bankoff,* Student Associate A.I.M.E.

(New York Meeting, February 1942)

Ince[1] proposed that electrostatic attraction between oppositely charged particles was responsible for slime-coating. Del Giudice[2] postulated the metathetic formation of a cementing compound. Wark[3] suggested that prevention of slime-coating was related to peptization of the slime.

In the experiments of Table 1 the technique employed was essentially that of del Giudice, with the difference that the particle was examined under a film of water rather than dry.[4] In general, the polished particle was agitated for 2 min. in a pulp consisting of 12.5 grams of the slime (−200 mesh) and 50 c.c. of distilled water, washed in three baths of distilled water, and examined at 220 diameters with the Leitz Ultropak. Observations of Brownian movement were made at 850 diameters with dark-field illumination.

Table 1.—*Experiments with Inorganic Electrolytes (Old Technique)*

Test	Particle	Preconditioning, Mg. per Liter	Slime	Treatment	Coating	Brownian Movement
a	Galena	None	Calcite	None[a]	Heavy	None
b	Galena	None	Calcite	2 lb. per ton sodium silicate	Very light	Violent
c	Galena	200 sodium silicate	Calcite	None	Heavy	None
d	Galena	200 KEtX	Calcite	None	Heavy	None
e	Galena	200 Na_2HPO_4	Calcite	None	Heavy	None
f	Galena	200 $K_2Cr_2O_7$	Calcite	None	Heavy	None
g	Galena	200 $Na_2C_2O_4$	Calcite	None	Heavy	None
h	Galena	None	Calcite	200 mg. per liter $Na_2C_2O_4$	Light	Medium
i	Galena	None	Calcite	Pulp from *h* filtered, washed with distilled water	Light	Medium
j	Galena	None	Quartz	None	Fairly light	Violent
k	Galena	100 KEtX	Quartz	None	Contact angle after sliming = 63°	Violent
l	Galena	None	Quartz	2.5 lb. per ton lime	Heavy	None
m	Galena	100 KEtX	Quartz	2.5 lb. per ton lime	Contact angle after sliming = 0°	None

[a] "None" signifies that the slime was untreated except for preliminary washing with alcohol and distilled water.

Several of these experiments do not jibe with the metathesis theory. According to published solubilities,[5,6] the lead salts of the anions in experiments *c* through *g* are more insoluble than lead carbonate, therefore should prevent the formation of lead carbonate on the galena surface. In experiment *i* the calcium oxalate, which presumably was on the calcite surfaces, should have permitted the formation of a lead carbonate cement by virtue of its greater solubility. Experiments *j* through *m* show that the addition of lime accelerates the deposition of quartz onto a galena surface. The writer is unable to postulate a satisfactory cementing compound, according to the metathesis theory, to explain this effect.

These experiments led to the conclusion that the metathetic formation of a cement-

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[1] References are at the end of the paper.

ing compound is not a necessary condition for slime-coating. They are sufficient in themselves to cause strong doubt as to whether metathetic surface compounds ever act as cements for slime adsorption.

calcite slime. The explanation lies in the fact that they are all salts of weak acids, which hydrolyze in water, leaving the solution somewhat alkaline. Hence the hydrolysis of the anchored carbonate ion

TABLE 2.—*Evidence in Support of Condition A*

Test	Particle	Treatment, Mg. per Liter	Slime	Treatment	Contact Angle[b]		Remarks
					Before	After	
1	Galena	Finger-greased	Quartz	Conditioned 3 min. in 1 lb. per ton Amine,[a] filtered, replaced with distilled water	90°	75°, 0°	Very heavy
2	Galena	200 Na oleate	Quartz	As above	85°	0°, 0°, 65°	Very heavy
3	Galena	200 KEtX	Quartz	As above	58°	62°	Very heavy in spots, fair in others
4	Galena	200 Na oleate	Calcite	1 lb. per ton Na oleate	90°	0°	Very heavy
5	Galena	200 KEtX	Quartz	As in 3 above	63°	0°	Fairly uniform, heavy, fine coat

[a] Armour amine AMAC-1180-C, guaranteed 93 per cent a mixture of octadecyl and hexadecyl amines. Solution prepared by dissolving the amine in dilute hydrochloric acid.

[b] When the contact angle differed widely over different points on the surface, all such readings are given.

The most satisfactory correlation of the observed facts has been obtained by postulating that the conditions controlling slime-coating are the same as those controlling flocculation. Brownian movement invariably accompanies dispersion, and the lack of Brownian movement ordinarily is accompanied by flocculation. According to this postulate, therefore, slime adsorption will occur when neither surface is in a state conducive to the Brownian movement. Galena is reported to be somewhat flocculated, and substantially at rest in distilled water.[7] Hence slime particles will be adsorbed on galena if their surfaces are not in a condition favorable to Brownian movement. On the other hand, they will be less strongly adsorbed, if at all, if their surface ionization is sufficient to produce Brownian movement. These statements agree with the observed facts.

Alkali chromates, carbonates, phosphates, oxalates, tungstates, or silicates in a calcite pulp, in concentrations of 100 mg. per liter, were found to reduce or prevent coating on a galena surface.[2] These electrolytes all induce Brownian movement in a on the calcite surfaces is repressed, increasing the charge on the particles, and, therefore, dispersing them more effectively. Lead acetate, which also inhibits coating of galena by calcite,[2] likewise disperses the calcite slime.

Ince[1] reported that lime increased coating of galena and sphalerite by Neihart slime, and also flocculated the slime. Since neither galena nor sphalerite show Brownian movement in distilled water,[7] we should expect flocculation of the slime, signifying repression of the surface ionization and consequent loss of Brownian motion, to be accompanied by an increase in slime-coating. Similar results were reported for galena and sphalerite particles in quartz slimes, with and without added lime.

All in all this is a remarkable correlation of the published and new data on slime-coatings with the new hypothesis.

EFFECT OF ORGANIC COMPOUNDS

In Tables 2 and 3 are summarized the results of a number of experiments that support the following conclusion: If (A) the surfaces of two mineral particles are made

water-repellent, or (*B*) if one particle is coated with a substance that is known to react with the other surface to make it water-repellent, and the two particles are brought into contact under water, they will adhere to each other. The general technique in performing these tests was the same as that already described, except that the particle was conditioned in the slime pulp for 3 min. instead of 2 min., after which it was washed under a moderately strong stream of tap water for 45 sec., instead of being washed in three baths of distilled water. After this treatment it was rinsed with running distilled water for a few seconds, and then transferred to a shallow dish of distilled water for microscopic examination.

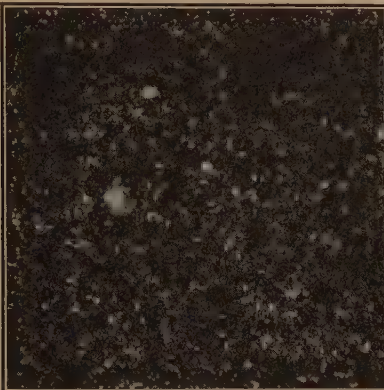

FIG. 1.—GALENA CONDITIONED 3 MINUTES IN 20 PER CENT CALCITE SLIME, WASHED IN THREE BATHS OF DISTILLED WATER AND DRIED. ULTROPAK. × 220.

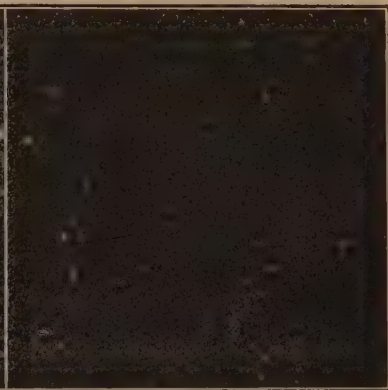

FIG. 2.—GALENA CONDITIONED AS IN FIG. 1, BUT WASHED UNDER STRONG STREAM OF TAP WATER. ULTROPAK. × 220.

In many cases, this change in the method of washing the particle had the effect of reducing coating markedly (compare Figs. 1 and 2).

Table 2 shows that a heavy slime-coating was produced whenever both the

TABLE 3.—*Evidence in Support of Condition B*

Test	Particle	Treatment, Mg. per Liter	Slime	Treatment	Contact Angle		Remarks
					Before	After	
6	Galena	200 Na oleate	Quartz	None	90°	90°	Almost no slime
7	Galena	200 KEtX	Quartz	None	65°	60°	Very light slime
8	Galena	200 KEtX	Calcite	None	60°	40°	Fairly light slime
9	Galena	200 Na oleate	Calcite	None	90°	0° (10 sec.) 75° (30 sec.)	Heavily coated
10	Galena	200 KEtX	Galena	None	56°	0°	Fairly heavily coated
11	Hematite	130 amine	Hematite	None	85°		Heavy
12	Hematite	130 amine	Quartz	None	85°		Heavy
13	Quartz	130 amine	Quartz	None	85°		Heavy
14	Quartz	130 amine	Hematite	None	88°		Very heavy
15	Hematite	None	Hematite	1 lb. per ton amine	0°		Fairly heavy
16	Hematite	125 oleic acid	Quartz	None	55°		Almost none
17	Hematite	125 oleic acid	Hematite	None	85°		Fairly heavy
18	Hematite	None	Hematite	2 lb. per ton oleic acid, filtered, washed with distilled water	0°		Fair slime Fine

polished surface and slime particle were made water-repellent. The results of a series of control experiments with clean, polished mineral surfaces and untreated slimes, using the changed technique, are given in Table 4. Heavy coating was induced whether the water-repellency was due to potassium ethyl xanthate, sodium oleate, oleic acid, Armour amine AMAC-1180-C, or finger grease, indicating that the phenomenon is independent of the nature of the substance inducing water-repellency.

TABLE 4.—*Experiments with Clean, Polished Surfaces and Untreated Slimes*

Test	Particle	Slime	Appearance
19	Hematite	Hematite	Virtually no slime
20	Hematite	Quartz	Virtually no slime
21	Quartz	Hematite	Virtually no slime
22	Galena	Calcite	Very little slime
23	Galena	Galena	Very little slime
24	Galena	Quartz	Virtually no slime

Besides being strongly adsorbed on the polished surface, the slime was heavily flocculated in each of the tests in Table 2. This indicates once again that there is a strong parallelism between slime-coating and flocculation.

The experiment with a galena surface preconditioned in potassium ethyl xanthate solution and a quartz slime conditioned in an acidified amine solution (both of which presumably were true solutions) demonstrates that the same degree of slime-particle adsorption is obtained even with monomolecular water-repellent coatings.

A finger-greased galena surface, of which the contact angle was 75°, was conditioned in a quartz slime for 30 sec. and then washed under the tap for 10 sec. The surface was fairly heavily slimed, but few of the particles were larger than 2 microns, and the contact angle after sliming was still 75°. The experiment was then repeated, except that a galena slime was substituted for the quartz slime. The surface was very heavily slimed with particles ranging up to 15 microns, and the contact angle was reduced from 70° to 0°. A galena surface is more easily wet,[8] hence rendered water-repellent, by finger grease than by quartz, and in accordance with condition *B*, galena is adsorbed at a grease-contaminated surface more readily than is quartz.

The idea that particles not wet by the liquid medium will stick together if collision occurs is not new. Fearnley[9] and Buzagh[10] arrived at similar conclusions. Taggart et al.[7] observed that the anchorage of a water-repelling film on particle surfaces induced flocculation, but indirectly postulated that such flocculation was similar to that produced by inorganic reagents. In both cases it was said that an insoluble, un-ionized film was formed on the surface, which lessened the repulsive forces between the particles and stopped Brownian movement.

If soluble collectors induce flocculation solely because they produce an insoluble, un-ionized surface, it would be expected that the flocculating power of different reagents for a given mineral would be only a function of the insolubility of the surface film formed; but this is not true. Galena conditioned in potassium dichromate cannot be floated subsequently with potassium ethyl xanthate, indicating that the lead dichromate film is more insoluble than the lead ethyl xanthate film. Both these reagents induce flocculation. However, if a polished galena particle is conditioned for 2 min. in a galena pulp to which has been added 8 lb. of potassium dichromate per ton, and is then washed under a strong stream of tap water for 10 sec., virtually no slime is seen on the surface at a magnification of 220. If the procedure is repeated with a galena slime that has been treated with 1 lb. of potassium ethyl xanthate per ton, a heavy coating appears.

In fact, the general observation was that the slime-coatings produced in the presence of organic reagents were far more strongly

adherent than coatings produced in their absence.

Effect of Gelatin

The addition of gelatin, in concentrations as low as 100 mg. per liter, had a remarkable effect in accelerating slime-coatings. The details are shown in Table 5. The nature of the polished surface or of the slime particles seemed to make no great difference, nor did it matter whether the polished surface was preconditioned in the gelatin solution or the gelatin was added directly to the slime pulp. Dense coatings were produced in every case. In the experiments in which the gelatin was added to the slime pulp, it induced heavy flocculation as well as slime-coatings.

A gelatin-treated surface was coated by amine-treated quartz particles, indicating that water-repellent surfaces are adsorbed by gelatin as well as water-avid surfaces. Further evidence on this score is the following: A galena particle conditioned in 200 mg. per liter gelatin solution was unaffected by subsequent conditioning in 200 mg. per liter potassium ethyl xanthate solution, as shown by the fact that the contact angle remained at the characteristic gelatin angle of less than 30° (clinging contact). However, if the procedure was reversed, the angle was reduced from 65° to 44°, indicating a fair amount of adsorption of the gelatin at the surface.

Summary

A general hypothesis is proposed. It is stated that the conditions that control the flocculation of mineral particles in water are the same as those that control slime-coating. The method of proof is indirect. A number of different conditions are examined where slime-coating is induced and where it is prevented; and it is shown in every case that flocculation follows an exactly analogous course.

Evidence is presented to the effect that:

1. Slime-coating is inhibited whenever the slime-particle surfaces are ionized sufficiently for Brownian movement to occur, and is facilitated when they are not.

2. Both slime-coating and flocculation are further aided by water-repellent surfaces on the adhering particles. It is found that adhesion occurs when one surface is coated with a substance that is known to react with the other surface to make it water-repellent, or when both surfaces are water-repellent.

3. Gelatin accelerates both slime-coating and flocculation tremendously.

Acknowledgment

Grateful acknowledgment is made to Prof. Arthur F. Taggart and Mr. Nathaniel Arbiter for their valuable advice and assistance in this work.

Table 5.—*Effect of Gelatin*

Test	Particle	Treatment, Mg. per Liter	Slime	Treatment	Contact Angle Before	Contact Angle After	Remarks
25	Pyrite	100 gelatin	Sphalerite	None	Cling	0°	Heavy slime
26	Hematite	100 gelatin	Hematite	None	Cling	0°	Very heavy. 75 per cent of field covered
27	Quartz	100 gelatin	Hematite	None	Cling	0°	Very heavy
28	Galena	100 gelatin	Calcite	None	Cling	0°	Heavy
29	Galena	100 gelatin	Galena	None	Cling	0°	Very heavy
30	Galena	100 gelatin	Quartz	None	Cling	0°	Very heavy
31	Galena	100 gelatin	Sphalerite	None	Cling	0°	Heavy
32	Galena	100 gelatin	Pyrite	None	Cling	0°	Fairly heavy
33	Galena	100 gelatin	Quartz	Treated with amine as in 1	Cling	0°	Heavy
34	Hematite	Clean	Hematite	2 lb. per ton gelatin	0°	0°	Very heavy
35	Quartz	Clean	Hematite	2 lb. per ton gelatin	0°	0°	Fairly heavy

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The Mechanism of Slime-coating

By Shiou-Chuan Sun,* Junior Member A.I.M.E.

(New York Meeting, February 1943)

There are several postulations for the mechanism of slime-coating. Ince[1] proposed the electrostatic hypothesis, del Giudice[2] suggested the chemical theory; Bankoff[3] reported that slime-coating is inhibited whenever the surfaces of the slime particles are ionized sufficiently for Brownian movement to occur, and is facilitated when they are not. These postulations were examined by experiments involving the Burton tube, the cataphoretic cell, and slime-coating. The techniques involved will be described elsewhere in this paper.

The electrostatic hypothesis, supported by migration experiments in a Burton tube, was abandoned by del Giudice.[2] Experiments of mineral migration in a Burton tube, after the idea of Ince with modification, indicated that this apparatus was inefficient for accurate observation. A cataphoretic cell was constructed. The experimental data, as shown in this paper, enable the correlation and advancing of this hypothesis.

Del Giudice's chemical hypothesis considered that slime particles are attached to the galena surfaces by a cement formed as the result of a metathetical reaction between the sulphide and the slime particles. Applying this hypothesis, the calcite slime particles were cemented to the galena surface by the formation of a cementing lead carbonate. In a similar way, lead silicate was considered as the cement of quartz slime coating on galena. Experiments verifying this hypothesis were carried out according to the following procedure: Fresh mineral particles selected from the central part of the broken lumps and examined first under a microscope for ascertaining a clean surface were placed in a small beaker (1.5-in. dia. by 2.5 in. deep) half filled with one kind of slime pulp. The dilution of all slime pulp was 4:1 and the particle size of slime was minus 400-mesh. After agitation for 5 min. the specimen was removed by a sharp-end tong and washed gently three times in three separate distilled water baths. It was dried before a small electrical fan and examined immediately under a microscope. A photograph was made of the observation.

With reference to the published solubilities,[4] several of these experiments, which contradict the chemical hypothesis, are listed as following:

1. The coating of fluorite slime on the galena surface was light but the coating of galena slime on the fluorite surface was heavy. In a similar way, the coating of quartz slime on the galena was fairly light, but the coating of galena slime on the quartz particle was heavy. According to the chemical hypothesis, using the same constituents and regardless of which is the slime, the density of the coating should be expected to be the same. This, however, as shown above, is not true. Moreover, fluorite (CaF_2), having a solubility of 16 mg.[4a] per liter of water, should not be

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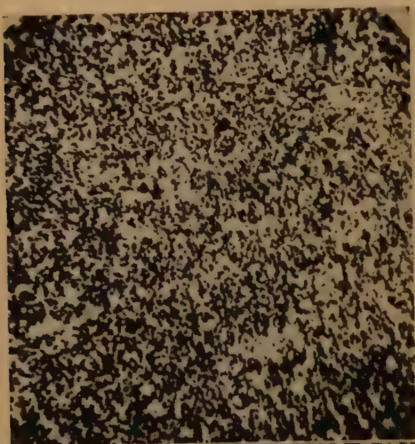
[1] References are at the end of the paper.

coated heavily by galena slime because it is less soluble than lead fluoride (640 mg. per liter of water), and also less soluble than calcium sulphate (1790 mg. per liter of water).

1


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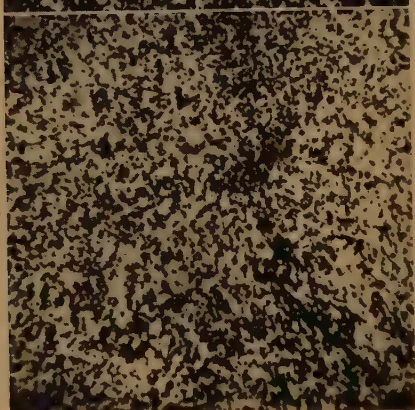

FIG. 1.—FLUORITE PARTICLE COATED BY QUARTZ SLIME IN DISTILLED WATER. × 300.
FIG. 2.—CHALCOPYRITE PARTICLE COATED BY QUARTZ SLIME IN DISTILLED WATER. × 300.

2. Barite ($BaSO_4$), having a solubility of 2.3 mg.[4b] per liter of water, is less soluble than lead sulphate (42.5 mg. per liter of water). However, the coating of barite slime on galena was fairly heavy.

3. The coating of galena slime on the galena particle, and the coating of calcite slime on calcite surfaces, cannot be explained by chemical hypothesis. The galena and calcite specimens were selected from the central parts of clean lumps. The slime was prepared by grinding the clean particles in a clean agate mortar. The chemical analyses of the samples used in the experiments are given as:

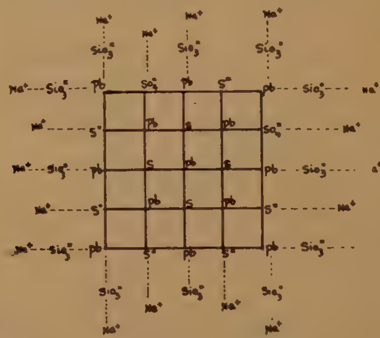

FIG. 3.—INDICATION THAT SODIUM SILICATE SOLUTION CAN INDUCE NEGATIVE CHARGE ON GALENA SURFACES.

galena, 99.8 per cent PbS (trace Fe, Ag); calcite, 99.3 per cent $CaCO_3$ (trace Fe, $CaSO_4$).

In a dilution of 4:1, the slime pulps were agitated 30 min. in distilled water by an electrical agitator. Chemical analysis of the filtrates are given as: galena filtrate, SO_4, 9.56 mg. per liter, SO_3, 5.15; Pb, 35.1 (no trace of Fe, Ag); calcite filtrate, Ca, 25.9 mg. per liter, CO_3, 39.3 (trace of SO_4).

These analytical data show that these coatings are probably not due to the presence of impurities but may be due to some other cause beyond the chemical hypothesis.

Brownian movement, as reported by Bankoff,[3] can indicate the magnitude of surface ionization but cannot show the sign of ionic charge, therefore it is an incomplete guide to the mechanism of slime-coating. This can be exemplified by the following observations. Quartz slime has a violent Brownian movement in distilled water. Fluorite also has a violent Brownian movement in distilled water. According to Bankoff's postulation, there should be a light slime-coating

between quartz and fluorite. On the contrary, the coating of quartz slime on fluorite and on chalcopyrite was heavy. These are shown in Figs. 1 and 2, respectively.

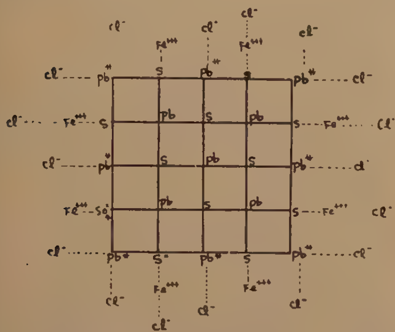

FIG. 4.—INDICATION OF CHANGE TO POSITIVE CHARGE BY CHEMICAL AGENTS.

These experiments led to the belief that chemical formation of a cementing compound is not necessary for slime-coating. The Burton tube is inefficient for furnishing the necessary data supporting the electrostatic hypothesis. For example, in distilled water, galena, sphalerite and quartz all migrated toward the positive electrode; yet in distilled water sphalerite slime and quartz slime coat galena readily. The idea of Brownian movement has its limitations and is not an all-inclusive method. In view of these facts, it was decided to attack the problem from the standpoint of magnitude and sign of charge of mineral surfaces. A cataphoretic cell was used for the measurement.

IONIC HYPOTHESIS OF SLIME-COATING

The satisfactory correlation of the following experimental data has been obtained by contending that slime-coating is controlled by the sign and magnitude of the ionic charge of particle and slime. Slime-coating is facilitated by the mutual attraction between the oppositely charged surfaces of particle and slime. Slime-coating is inhibited whenever the ionic repulsion between particle and slime occurs. Since the two contacting surfaces are both rigid and rough, the binding force of slime-coating is much weaker than that of a true chemical combination. This agrees with the phenomenon that slime-coating

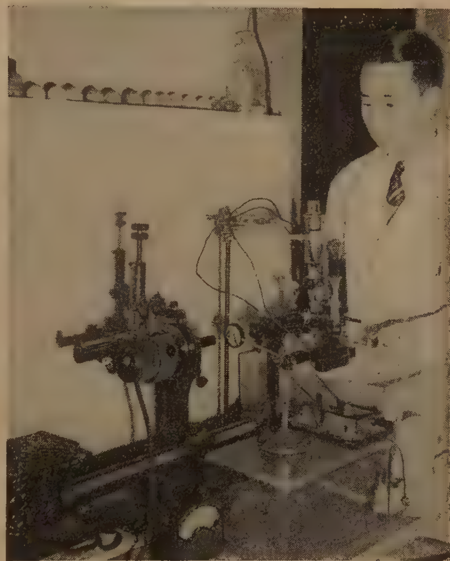

FIG. 5.—CATAPHORETIC CELL.

can be washed away by a strong stream of water or removed easily by a piece of cotton. From the experimental data, as shown in Tables 1 to 4 the relationship between the zeta potential and slime-coating is established as follows:

1. Slime-coating is heavy when both slime and particle have a high zeta potential and are opposite in sign.

2. Slime-coating is light when the zeta potential of slime is high and the zeta potential of the particle is low.

3. Slime-coating is heavy when the zeta potential of slime is low.

4. Slime-coating is prevented when the zeta potential of slime and of the particle are both high and alike in sign.

The idea that mineral particles are ionically or electrically charged in water or in aqueous solution is not new. Quincke discovered the phenomenon of cataphoresis

in 1850. Gouy's double-layer theory[5] is still generally accepted. According to the idea of H. B. Weiser,[6] a double layer may be formed at the surface of colloidal particles by: (1) preferential adsorption of or chemical reaction between particle and electrolyte is governed by the Fajans-Hahn law, which states that such ions are preferentially attracted as can be incorporated into the lattice structure.

TABLE 1.—*Slime-coating Heavy When Both Slime and Particle Have High Zeta Potential and Are Opposite in Sign*

Test No.	Galena: Type	Galena: Zeta Potential, Mv.	Slime: Kind	Slime: Zeta Potential, Mv.	Coating
1	Galena	+28.8, (−16.9)[a]	Fluorite	+91.7	Light
2	Galena soaked 72 hr. in 175 grams per liter Na_2SiO_3, washed with distilled water	−32.3	Fluorite	+91.7	Medium
3	Galena soaked 4 min. in 1167.5 mg. per liter $FeCl_3$, washed with distilled water	+50.8	Quartz	−41.4	Heavy
4	Chalcopyrite	+57.3	Quartz	−41.4	Heavy
5	Quartz	−41.4	Fluorite	+91.7	Heavy
6	Fluorite	+91.7	Quartz	−41.4	Heavy

[a] The zeta potentials shown in parentheses in Tables 1 and 2 represent the zeta potential of the negatively charged galena particle in distilled water. See pp. 480 et. seq. and Table 5.

TABLE 2.—*Slime-coating Light When Zeta Potential of Slime Is High and Zeta Potential of Particle Is Low*

Test No.	Particle: Kind	Particle: Zeta Potential, Mv.	Slime: Kind	Slime: Zeta Potential, Mv.	Coating
1	Galena	+28.8 (−16.9)	Quartz soaked 30 min. in 0.4 grams per liter Na_2SiO_3, washed with distilled water	−50.4	Light
2	Galena	+28.8 (−16.9)	Quartz soaked 30 min. in 0.4 grams per liter Na_3PO_4, washed with distilled water	−50.2	Light
3	Galena	+28.8 (−16.9)	Quartz	−41.4	Fairly light
4	Galena	+28.8 (−16.9)	Fluorite	+91.7	Light
5	Galena	+28.8 (−16.9)	Sphalerite soaked 3 hr. in 0.6 grams per liter $Al(NO_3)_3$, washed with distilled water	+57.6	Light
6	Calcite	−22.9 (+21.5)	Quartz	−41.4	Light

one ion of an electrolyte in which the particle is suspended or (2) by direct ionization of some of the surface molecules. In the absence of one or the other of these phenomena, a double layer may result from selective adsorption of H^+ or OH^- ions from water. The reason for the direct ionization and preferential adsorption of the particle in liquid finds its explanation in the postulation of I. Langmuir[7] and E. A. Hauser,[8] that mineral surfaces are unsaturated. Sorption

Furthermore, there always will be a trend to form the least soluble matter. One point appears certain, that chemical reagents can alternate the magnitude and sign of ionic charge as shown in the experimental data of a cataphoretic cell. Fig. 3 indicates that a sodium silicate solution can induce a negative charge on galena surfaces. On the other hand, ferric chloride, aluminum nitrate, and some other chemicals, can change the surface charge to positive, as shown in Fig. 4.

For the purposes of simplicity, the magnitude and sign of ionic charge can be represented by the term "zeta potential."[6,8] In this paper, the zeta potential represents the potential gradient between the surface of a solid particle in suspension and the liquid adjacent to the particle, and is the result of the tendency of ions on the surface of the particle and within the liquid boundary layer to establish a condition of equilibrium. A high positive zeta potential means that the mineral surface is dominated by cations, a high negative zeta potential indicates the domination of anions, and a low zeta potential indicates that the surface is influenced almost equally by cations and anions.

TABLE 3.—*Slime-coating Heavy When Zeta Potential of Slime Is Low*

Test No.	Particle		Slime		Coating
	Kind	Zeta Potential, Mv.	Kind	Zeta Potential, Mv.	
7	Calcite	−22.9 +21.5	Fluorite	+91.7	Fairly light
8	Galena	+28.8 −16.9	Galena	+28.8 −16.9	Heavy
9	Galena	+28.8 −16.9	Gypsum	−18.3	Heavy
10	Galena	+28.8 −16.9	Talc	−19.9	Heavy
11	Galena	+28.8 −16.9	Witherite	−22.4	Heavy
12	Galena	+28.8 −16.9	Calcite	−22.9 +21.5	Heavy
13	Galena	+28.8 −16.9	Kaolinite	−24.2	Heavy
14	Galena	+28.8 −16.9	Fluorite soaked 30 min. in 200 mg. per liter Na_2SiO_3, washed with distilled water	−22.3	Heavy
15	Galena	+28.8 −16.9	Quartz soaked 20 min. in 1.2 grams per liter $FeCl_3$, washed with distilled water	+29.5 −28.3	Heavy
16	Galena	+28.8 −16.9	Barite	−38.8	Fairly heavy
17	Calcite	−22.9 +21.5	Calcite	−22.9 +21.5	Heavy
18	Calcite	−22.9 +21.5	Galena	+28.8 −16.9	Heavy
19	Sphalerite	+42.4 −15.2	Calcite	−22.9 +21.5	Heavy
20	Fluorite	+91.7	Calcite	−22.9 +21.5	Heavy
21	Quartz	−41.4	Galena	+28.8 −16.9	Heavy

EXPERIMENTS USING A CATAPHORETIC CELL

The migration of mineral particles was first tried in a Burton tube, but this apparatus was found to be inefficient. A cataphoretic cell was built, after the idea of Sante Mattson[9] with modification. By use of this cell, the mobility of a single particle and its ionic charge can be measured. Brownian movement can also be roughly observed. The sign of charge can be indicated by the direction of migration and the magnitude of the charge can be calculated from the mobility of the particle.

The cell, as shown in Fig. 5, consists of a thick-walled tube of 1.53 mm. inside diameter and 13.3 cm. long, terminating in two larger tubes. At the central part, the tube is ground down to within 0.2 mm. of the inner wall. At right angle to the ground surface, on the side of the tube where the light is to enter, another plane surface is similarly made. The large platinum electrodes are placed in the large tubes, at each end. The particles are illuminated on the principle of the ultra-

microscope, the source of light being an electric arc. A collecting lens is attached to the arc lamp. After the collecting lens, the convergent light is focused on an adjustable slit. The position of the objective can be adjusted by moving it back and forth to find the place where the image of the slit is focused inside the glass tube directly under the microscope. By means of a commutator, the current can be reversed. The microscope used in this apparatus is a Bausch and Lomb with an 8-mm. objective (N.A. = 0.5) of 1.6-mm. working distance. The eyepiece is 10X and is provided with a scale which covers 0.5 mm. The potential difference at the electrodes deviated slightly from 205 volts. The potential gradient is 15.41 volts per centimeter. The observed velocity of a particle expressed in microns per second in a gradient of one volt per centimeter is equal to $500/15.41S$, where S is the time in seconds required by a particle to cover the scale (0.5 mm.). The particle movement is measured in both directions with a stop watch.

The true velocity of migration is in the annular layer. After the theoretical deductions of Smoluchowski,[10] the annular layer of this cell was calculated to be 0.2142 mm. from the inside wall surface. The experimental data of mineral migra-

TABLE 4.—*Slime-coating Prevented When Zeta Potential of Slime and of Particle Are Both High and Alike in Sign (Very Light Slime-coating)*

Test No.	Reagent in Pulp, Mg. per Liter		Particle		Slime	
			Kind	Zeta Potential, Mv.	Kind	Zeta Potential, Mv.
1	Na_2SiO_3	250	Galena	−33	Sphalerite	−48
2		200	Galena	−30	Quartz	−80
3		400	Galena	−50	Calcite	−66
4		400	Galena	−50	Barite	−74
5		400	Galena	−50	Bentonite	−65
6		400	Galena	−50	Kaolinite	−57
7	Na_3PO_4	687	Galena	−50	Calcite	−35
8		100	Galena	−42	Quartz	−50
9		860	Galena	−50	Bentonite	−57
10		460	Galena	−50	Kaolinite	−50
11		200	Galena	−41	Sphalerite	−40
12	$Na_2C_2O_4$	300	Galena	−65	Calcite	−40
13		200	Galena	−65	Galena	−65
14		175	Galena	−60	Sphalerite	−65
15		175	Galena	−60	Quartz	−72
16	K_2CrO_4	100	Galena	−52	Quartz	−50
17		200	Galena	−62	Galena	−62
18		200	Galena	−62	Calcite	−38
19	$AL(NO_3)_3$	1,000	Galena	+58	Galena	+58
20		1,000	Galena	+58	Calcite	+78
21	$FeCL_3$	100	Galena	+66	Galena	+66
22		1,000	Galena	+66	Quartz	+115
23		700	Galena	+66	Calcite	+56
24	Amine (1180-C)	800	Galena	+186	Galena	+186
25		800	Galena	+186	Calcite	+198
26		800	Galena	+186	Quartz	+228
27		800	Galena	+186	Fluorite	+206
28	$Pb(C_2H_3O_2)_2$	780	Galena	+65	Galena	+65
29		780	Galena	+65	Quartz	+65
30		780	Galena	+65	Calcite	+50
31	Na_2CO_3	1810	Galena	−127	Galena	−127
32		1000	Galena	−102	Quartz	−122
33	Na_2WO_4	400	Galena	−66	Galena	−66
34		200	Galena	−50	Quartz	−92
35		200	Quartz	−92	Galena	−50
36	K-Am.-X	1300	Galena	−76	Quartz	−42
37	Na_2SiO_3	100	Fluorite	−48	Quartz	−66
38	Na_3PO_4	200	Fluorite	−40	Quartz	−60
39	Na_3PO_4	100	Quartz	−50	Galena	−52
40	Na_3PO_4	687	Calcite	−35	Galena	−50
41	Na_2SiO_3	400	Calcite	−66	Galena	−50

tion, as shown in Figs. 6 to 13 were measured in the annular layer and calculated according to the method used by Sante Mattson.[9] chloride solution, are dominated by cations. The quartz slime surfaces are dominated by anions, as indicated by the high negative zeta potential. The cationic surfaces

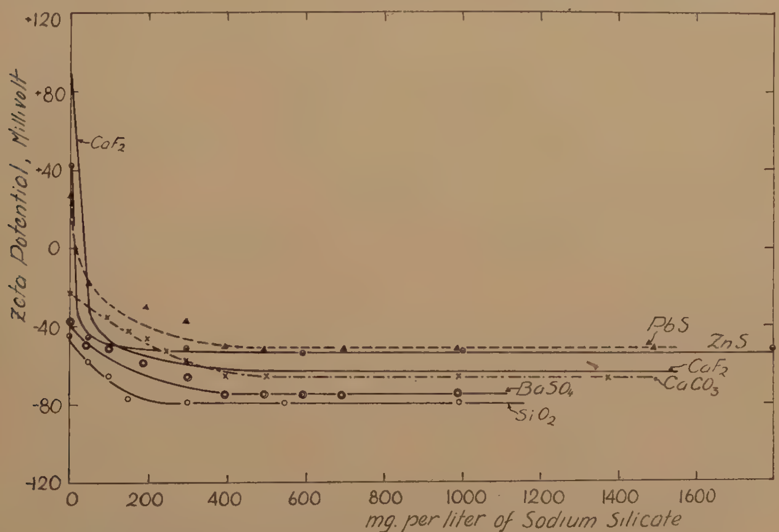

FIG. 6.—MIGRATION OF MINERALS IN SODIUM SILICATE SOLUTION.

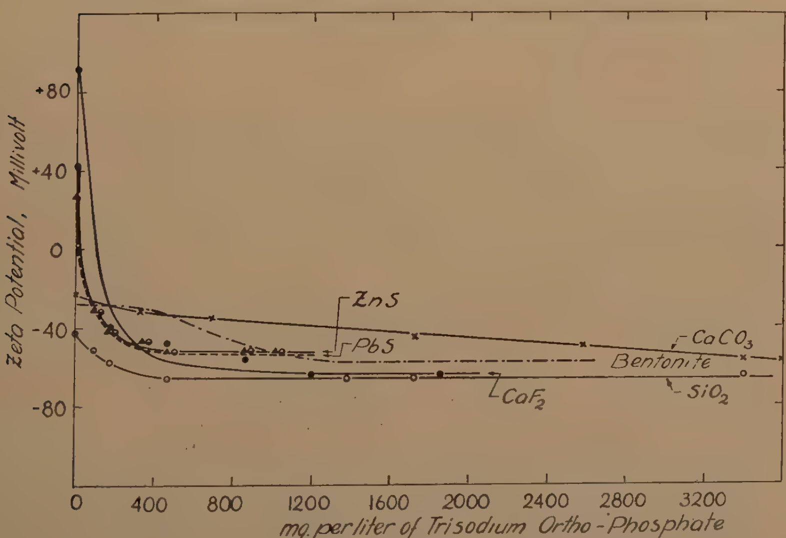

FIG. 7.—MIGRATION OF MINERALS IN TRISODIUM ORTHOPHOSPHATE SOLUTION.

DISCUSSION OF RESULTS

In Table 1, the particles of high positive zeta potential, such as fluorite, chalcopyrite, and galena preconditioned with ferric can make a good mutual attraction with the anionic surfaces and slime-coating is heavy. This is shown schematically in Figs. 14 and 15. The zeta potential of

quartz slime is high and must be accompanied by dispersion. Dispersed quartz slime is not favorable for slime-coating. However, in this case, the attraction is great enough to overcome this unfavorable condition.

Experiments of Table 2 indicate that slime-coating is light when the zeta potential of the particle is low and that of the slime is high. This is demonstrated in Figs. 16, 17 and 18. The combination between slime dominated by either cation or anion and particle with nearly equal influence of cation and anion is difficult to arrange because of the presence of repulsive force, as shown in Figs. 16 and 17.

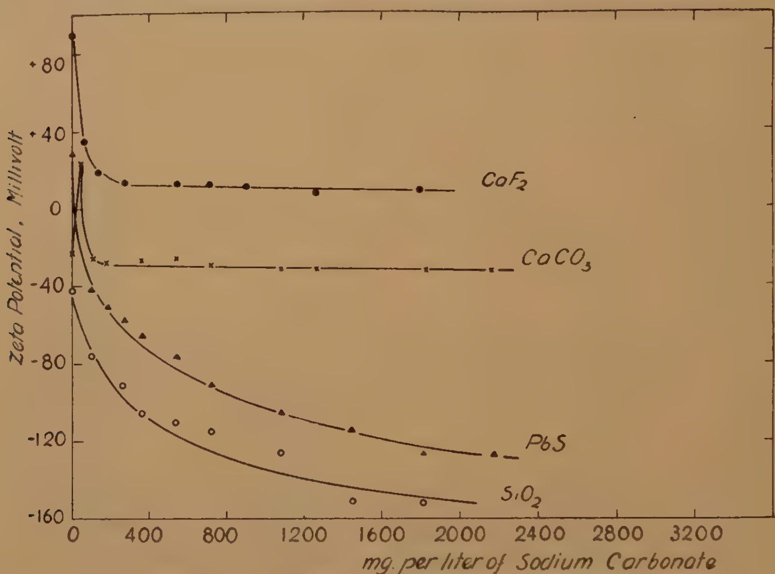

FIG. 8.—MIGRATION OF MINERALS IN SODIUM CARBONATE SOLUTION.

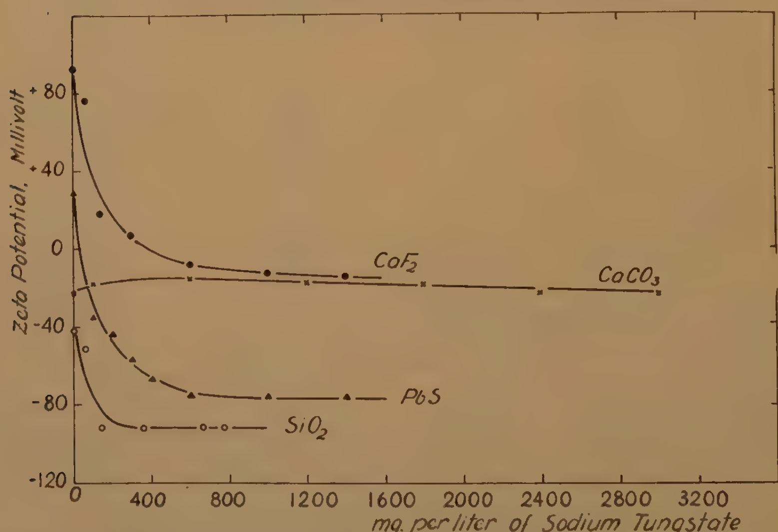

FIG. 9.—MIGRATION OF MINERALS IN SODIUM TUNGSTATE SOLUTION.

Besides, from Fig. 18, we can visualize that slime-particles of high zeta potential have thicker double layers and the repulsive force among themselves is large. In the pulp, the slime particles will be far apart from each other (dispersion). Slime particles coated on or near the particle surfaces will prevent the approaching of other slime-particles. Therefore, slime-coating should be light.

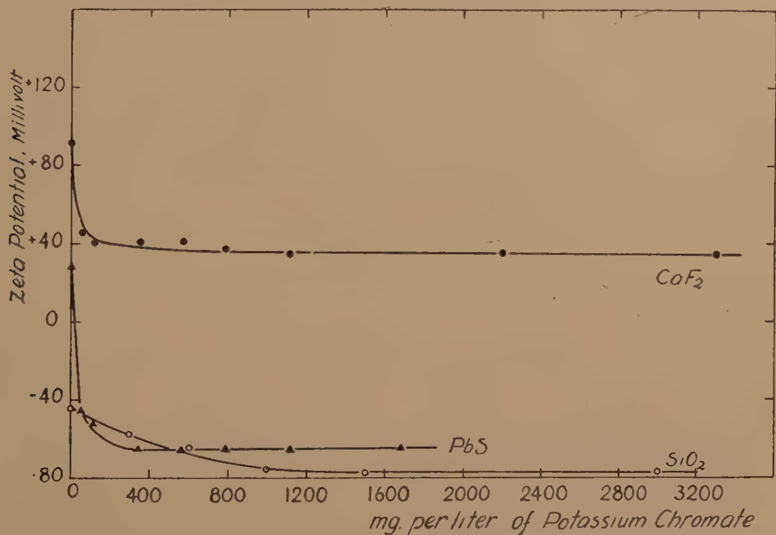

FIG. 10.—MIGRATION OF MINERALS IN POTASSIUM CHROMATE SOLUTION.

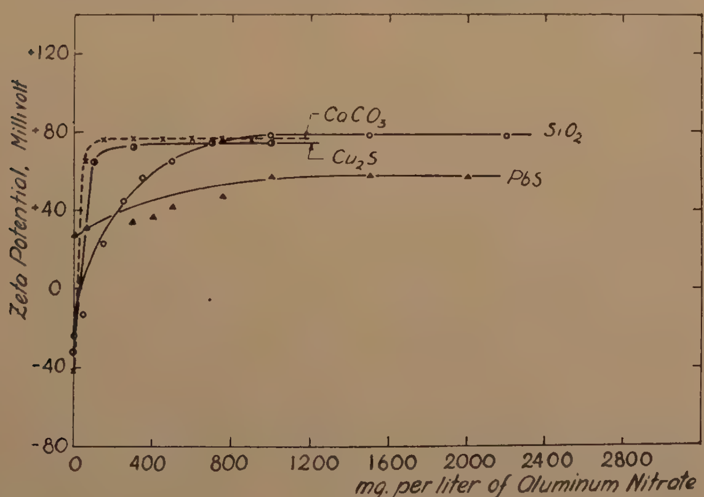

FIG. 11.—MIGRATION OF MINERALS IN ALUMINUM NITRATE SOLUTION.

Table 3 is the reverse of Table 2. Slimes with a low zeta potential invariably have a tendency to group themselves into small flocs, because the double layer is thin and the ionic attraction is easily arranged, as shown in Fig. 19. When one part of the floc is attached to the particle surface, the whole body of the floc will adhere.

The outstanding characteristic property of the low zeta potential slime, such as

calcite and galena, is the presence of both positive and negative particles. Since the surface is composed of an almost equal number of cations and anions, and the zeta potential is close to the isoelectric point, the sign of charge is easily disturbed.

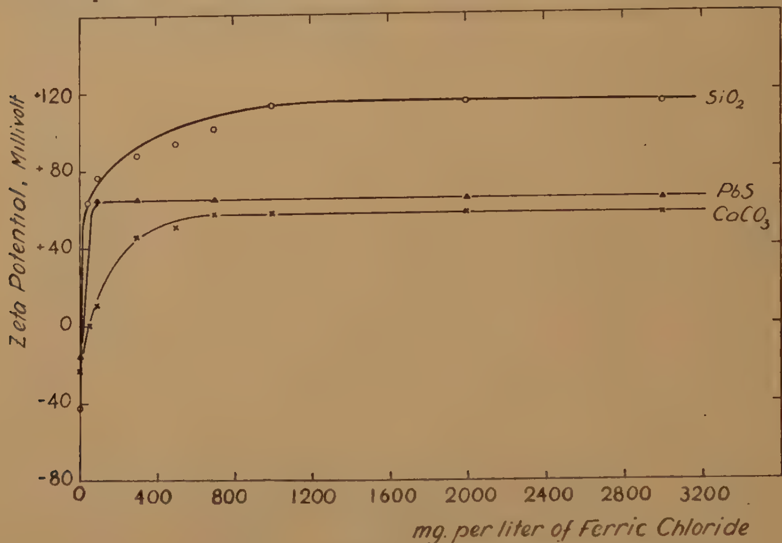

FIG. 12.—MIGRATION OF MINERALS IN FERRIC CHLORIDE SOLUTION.

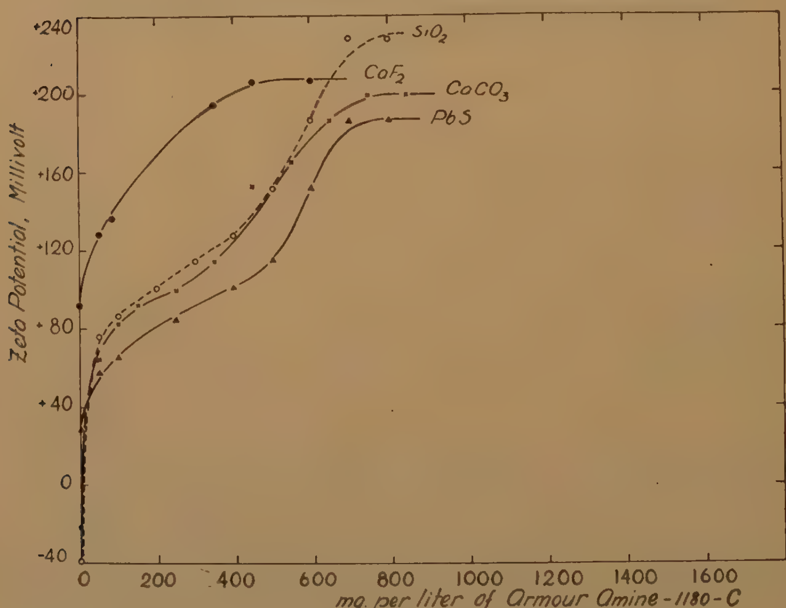

FIG. 13.—MIGRATION OF MINERALS IN ARMOUR AMINE 1180-C.

The disturbance may be caused by the losing of corners or uneven breakage through crushing and grinding or by the uneven preferential adsorption of different surfaces. Higher and lower positive particles and low negative particles were

found in the cataphoretic cell when using galena slime with distilled water; higher and lower negative particles and low positive particles were found when using calcite slime with distilled water. In the slimes of barite, talc, gypsum and kaolinite, very small traces of low positive particles were found. These can be confirmed indirectly by del Giudice's data.[2] He reported, in the observation of his bakelite cell, that: "With galena in distilled water and in 0.001-molar carbonate solution, with sphalerite in distilled water, and with calcite in distilled water, the results were doubtful." This may be due to the presence of both positive and negative particles. As said before, without the predetermination of the cell's annular layer, this phenomenon can perplex the observer. In a solution of 0.001-molar sodium carbonate, 25 to 30 per cent positive and 75 to 70 per cent negative galena slime particles were found in this cataphoretic cell. The presence of both positive and negative particles of minerals in distilled water is shown in Table 5.

The presence of both positive and negative particles in the slime makes possible its adhesion on any kind of particle surface. On a high negative particle surface, slime-coating may take place according to Fig. 20. For high positive particle surfaces, the arrangement is shown in Fig. 21. For low zeta potential surfaces, the coating is formed according to Fig. 22.

Table 4, the prevention of slime-coating, can easily be explained by the fact that both slime and particles are dominated by the same kind of ions; they will repel instead of attract each other. Moreover, the thick double layer and the dispersion of slime are not likewise favorable for slime-coating.

The following experimental results, which cannot be interpreted by the chemical theory but are adequately explained by the above postulations, are: (1) the coating of barite slime on galena; (2) galena slime on galena; (3) calcite slime on calcite; (4) light coating of fluorite slime on galena but heavy coating of galena slime on fluorite. The mechanism of experiments 1, 2 and 3 are shown in Figs. 23, 24 and 25, respectively. Experiment 4 can be visualized by the combination of Figs. 16, 18, 19 and 20.

TABLE 5.—*Presence of Both Positive and Negative Particles of Minerals in Distilled Water*

Minerals	Zeta Potential, Mv.	Percentage of Positive Particles
Fluorite (only positive particles)	+91.7	100
Quartz (only negative particles)	−41.4	0
Sphalerite	+42.5(−15.2)	50–60
Galena	+28.8(−16.9)	25–35
Barite	−38.8	Tr.
Calcite	−22.9(+21.5)	20–30
Kaolinite	−24.2	Tr.
Quartz soaked 20 min. in 1.2 grams per liter $FeCl_3$, washed with distilled water	+29.5(−28.3)	70–90

The experimental curves shown in Figs. 6 to 13 can be used to control slime-coating. The turning point of a curve represents the least amount of the reagent that can form the maximum ionic charge on the particle surfaces. With consideration of both the particle curve and slime curve, the least but still effective amount of reagent can be found to prevent slime-coating. The effective amount of inorganic reagents can be somewhat lower than the listed data in Table 4 but this is not true for organic reagents.

The cataphoretic curves can also be used to control dispersion and flocculation. There is a qualitative parallelism between dispersion and a large zeta potential, as there is between flocculation and a small zeta potential. Experiments indicated that a good dispersion can be expected when the zeta potential is above 40 mv. Organic reagents, such as sodium oleate and amines, flocculate certain minerals at a consider-

ably higher zeta potential; redispersion can be achieved only at a very high zeta potential by increasing the concentration.

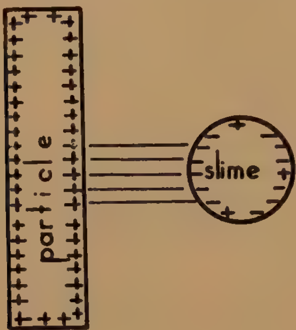

FIG. 14.

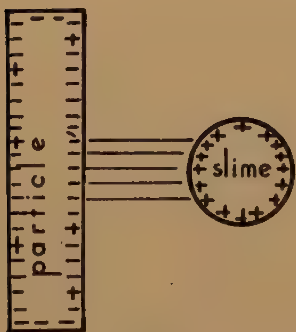

FIG. 15.

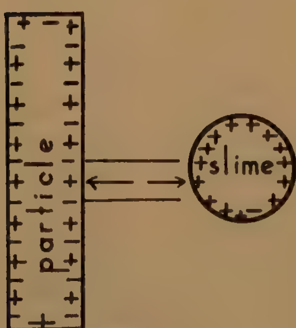

FIG. 16.

EFFECT OF SLIME PARTICLE SIZE

Different sizes of slime particles were prepared by a short-column hydraulic elutriator.[11] The results of experiments indicated that slime-coating is increased by the decrease in size of the slime particle. Above 400-mesh, the coating of quartz,

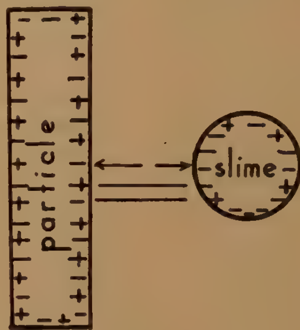

FIG. 17.

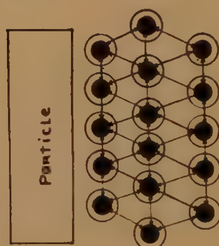

FIG. 18.

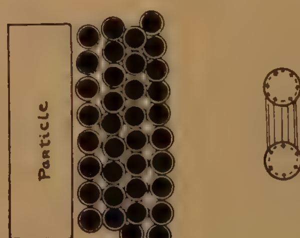

FIG. 19.

calcite, barite, and sphalerite slime on galena was very light. The coating was increased proportionally from minus 400-mesh to 2 microns. The finer slime produces a higher specific surface and more slime particles in the pulp. Distance between slime particles is shortened and the probability of collision is increased. The force of ionic attraction is capable of holding only the smaller slime particles but not strong enough to keep the bigger slime particles on the particle surface.

Effect of Agitation Period

The experimental data of slime-coating with different agitation periods indicated

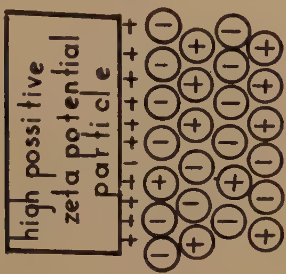

Fig. 20.

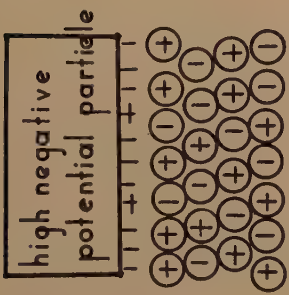

Fig. 21.

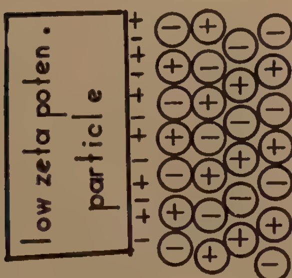

Fig. 22.

that the coating was increased by the increase of agitation. The probability of collision between particle and slime is increased by the longer time of agitation. Hence, slime-coating should be increased.

Conclusion

An ionic hypothesis is proposed. It is stated that slime-coating is controlled by the magnitude and the sign of the ionic charges of both particles and slimes. Mutual attraction induces slime-coating, and repulsion prevents slime-coating. Ionic charge of mineral surfaces, as represented

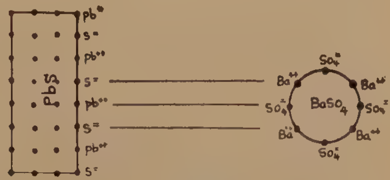

Fig. 23.

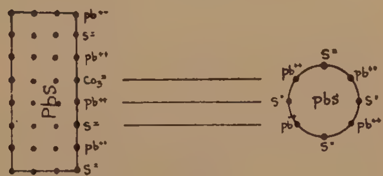

Fig. 24.

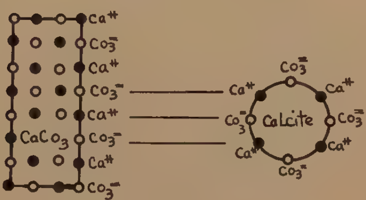

Fig. 25.

by the zeta potential, is determined by the domination of anions or cations and can be regulated by chemical reagents. The relationship between slime-coating and the zeta potential is established by the experimental data of the cataphoretic cell and of slime-coatings. The results of experiments can be summarized as follows:

1. Slime-coating is heavy when zeta potentials of particle and slime are both high but opposite in sign.

2. Slime-coating is light when the zeta potential of slime is high and that of the particle is low.

3. Slime-coating is heavy when the zeta potential of slime is low.

4. Slime-coating is prevented when the zeta potentials of slime and the particle are both high and alike in sign.

5. Slime-coating is facilitated by the decrease of slime-particle size when the other factors are kept constant.

6. Slime-coating is increased by the increase of the agitation period.

7. The least but still effective amount of inhibiting reagents for the prevention of slime-coating is shown on the turning points of the cataphoretic curves.

8. A high zeta potential is accompanied by Brownian movement and dispersion. A low zeta potential is associated with flocculation and shows no Brownian movement.

Acknowledgment

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The Nature of Dispersed Mineral in Flotation Pulps

By T. C. Fitt,* A. W. Thomas† and Arthur F. Taggart,‡ Member A.I.M.E.

(New York Meeting, February 1943)

It was noticed early by operators that high recoveries and flocculation of the sulphide minerals were closely correlated in agitation-froth flotation. Later, this readily visible flocculation was found almost invariably to be due to the binding together of the oiled sulphides by minute air bubbles, and is now known to be characteristic of pulps in which at least one of the mineral species is coated with an insoluble collecting oil.

In 1930 it was reported[1] that in differential flotation a correlation exists between Brownian movement of particles in the pulp and their nonfloatability, and that Brownian movement and dispersion are likewise correlated, whence it would appear to follow that dispersion of a mineral particle in a flotation pulp denotes a particle condition unfavorable to flotation.

In 1934 it was postulated[2] that Brownian movement, dispersion and nonfloatability all flow from the condition of the surface of the dispersed particles. It was also postulated at that time that the surfaces of dispersed particles are ionized, the ionized compound being a product of reaction between a component of the particle and a component of the pulp, of a mass solubility of the general order of a fraction of a milligram to 15 to 20 mg. per liter. It was then further asserted, and has since been confirmed by Bankoff,[3] that sulphides so conditioned with soluble collectors as to be floatable flocculate without the intervention of air bubbles as binding media.

The present paper comprises an inquiry into the nature of the bound and counter ions in zinc sulphide dispersed by sodium arsenate. A part of its findings was mentioned in the 1934 paper above cited.[2] The tests show that:

1. When powdered granular zinc sulphide is immersed in aqueous solutions of sodium arsenate, arsenate ion is abstracted from solution and anchored to the solid surface, while sulphate and lower sulphoxy ions are concurrently displaced therefrom in substantially stoichiometric quantities.

2. The ion counterbalancing the anchored arsenate ion is zinc.

3. Dispersion and Brownian movement of zinc sulphide are concurrently maxima at an arsenate-ion concentration lying between 0.01 and 0.001-molar sodium arsenate.

4. Any condition or reaction that tends to remove the zinc arsenate coating from the zinc sulphide particles decreases dispersion and Brownian movement concurrently.

Experimental

Three samples of zinc sulphide were used. Lot 1 was C.P. zinc sulphide, washed well beyond absence of chloride, as shown by test with silver ion. Lot 2 was prepared by precipitation from hydrochloric acid solution of arsenic-free zinc, and was washed as lot 1. Lot 3 was prepared by heating a portion of lot 2 in absence of air at 1200° to 1300°C. for one hour.[4]

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[1] References are at the end of the paper.

It was coarsely crystalline, pale brownish green, anisotropic. All grinding of samples was done in agate mortars. All reagents used were tested-purity reagent grades. All glassware was Pyrex except that soft-glass bottles were used for agitation of suspensions; test runs indicated no differences in results when Pyrex was substituted for soft-glass bottles. Standard methods of analysis were employed, chosen after tests had shown that determinations therewith departed not more than 3 per cent of compounded concentrations from such figures, and that the spread in results of duplicate analyses was not greater than 3 per cent of the determined values.

Ion Exchange

Experiment I.—Lots comprising 30 grams of zinc sulphide (for lot numbers see footnote to Table 1) and 300 ml. of 0.002-molar sodium arsenate solution, substantially filling a glass-stoppered soft-glass bottle, were rotated for periods of one hour or more at room temperature, in such a manner that the zinc sulphide particles were continually showered through the liquid. The suspension was then centrifuged for one hour at 1800 r.p.m. and radius of 21 cm. The supernatant liquid was still faintly turbid. Determinations made on 25-ml. aliquots of the supernatant liquid for sulphate, arsenate, sodium, and sulphoxy ions[6] are recorded in Table 1.

TABLE 1.—*Results of Experiment* I

Run No.	Original Solution, Mg. per Liter		Centrifugate			Arsenate Ion Removed by Zinc Sulphide, Mg. per 100 Grams ZnS			
						Total Mg. per 100 Gram ZnS	Accounted for as		
	Sodium Ion	Arsenate Ion	Sodium Ion, Mg. per Liter	Sodium Ion Removed by Zinc Sulphide, Mg. per 100 Gram ZnS	Arsenate Ion, Mg. per Liter		Sulphate Ion[a]	Sulphoxy Ions[b]	Unaccounted for
1	92.3	279.0	93.6	0.0	127.0	152.0	84.3		
2	99.2	277.7	97.6	1.6	178.8	98.9	52.9		
3	97.0	280.0	96.0	1.0	190.8	89.2	36.6	37.0	15.6
4	95.2	278.0	93.6	1.6	166.8	111.2	40.9	55.5	14.8
5	96.0	278.0	95.2	0.8	133.0	145.0	57.8	63.8	22.4

[a] $SO_4^{=}$ calculated to arsenate-ion basis.

[b] Knoll[5] showed that a similar filtrate obtained from galena by extraction with a solution of potassium ethyl xanthate, from which xanthate ion was completely removed, gave a white precipitate with $AgNO_3$, and that this precipitate rapidly turned brown and then black. This indicates thiosulphates or other thionates.[8] "Sulphoxy" ion figures above analytically measured as $SO_3^{=}$ are calculated to arsenate-ion basis.

[c] Runs 1 and 2 were performed with lot 2; runs 3, 4 and 5 with lot 1.

Experiment II.—Weighed quantities of powdered, washed lot 2 were placed in 300-ml. portions of distilled water in glass-stoppered bottles and equal quantities of the same lots were similarly placed in 300-ml. portions of 0.002-molar sodium arsenate solution. Both preparations were rotated for the same times at room temperature and then allowed to settle. The distilled-water suspensions settled completely in 5 min.; the sodium arsenate suspensions were dispersed. They were centrifuged for 30 min. to 1 hr. at 1800 r.p.m. and 21-cm. radius, to remove the bulk of the solid particles. Results of sulphate tests on supernatant liquids are shown in Table 2.

TABLE 2.—*Results of Experiment* II

Run No.	Time of Contact of Zinc Sulphide with Liquid, Hr.	Sulphate Ion[a] Displaced from Zinc Sulphide by Distilled Water, Mg. per 100 Grams ZnS	Sulphate Ion[a] Displaced from Zinc Sulphide by Sodium Arsenate Solution, Mg. per 100 Grams ZnS
1	1	16.1	53.8
2	2	10.1	45.2
3	24	8.4	44.1
4	24	21.2	59.0

[a] The samples for runs 2 and 3 were coarser than for runs 1 and 4.

Experiment III.—Portions of lot 3 weighing 25 grams were shaken with 100-ml. portions of aqueous sodium arsenate solutions of which the

concentrations ranged from 1 to 10^{-6} molar. The particles were allowed to settle and the solutions were decanted. The settled solids were separately washed by elutriation with tap water at such rising velocities that the larger zinc sulphide particles were suspended but not overflowed, while the smaller particles were overflowed and discarded. This treatment was followed by several washings of the granular residue with distilled water. The washed solid, after drying, was analyzed for arsenic. Arsenic tests were also run on the original material of lot 3.

Experiments were also run by shaking 25 grams of deslimed $-65 + 150$-mesh quartz with 100 ml. of 0.01-molar sodium arsenate solution for one hour, then washing for 15 min. by elutriation as above, then analyzing for arsenic. In another experiment, $-65 + 150$-mesh quartz was suspended in 100 ml. of 0.01-molar sodium arsenate solution and zinc sulphate solution was added until a precipitate of zinc arsenate appeared. The suspended arsenate was decanted from the quartz and the latter was washed by elutriation for 15 min., then analyzed for arsenic. The results of these experiments are shown in Table 3.

TABLE 3.—*Results of Experiment* III

Run No.	Arsenic on Crystalline Zinc Sulphide after Washing with Distilled Water, Mg.As per 100 Grams ZnS	Arsenic on Crystalline Zinc Sulphide after Treatment with Sodium Arsenate Solutions			
		Concentrated Sodium Arsenate Solution, Mols per Liter	Time of Contact with Sodium Arsenate Solution, Min.	Time of Washing with Running Water, Min.	Arsenic on Crystalline Zinc Sulphide, Mg.As per 100 Grams ZnS[a]
1	0.10	1.0	60	35	0.75
2	0.10	0.1	60	35	0.60
3	0.10	0.01	60	15	0.50
4	0.10	0.01	60	15	0.30
5	0.10	0.01	120	15	0.50
6	0.10	0.01	15	15	0.50
7	0.10	0.001	60	15	0.30
8	0.10	0.0001	60	15	0.25
9	0.10	0.00001	60	15	0.10
	Experiments with quartz in contact with sodium arsenate solution				
10	0.00	0.01	60	15	0.00
	Experiments precipitating zinc arsenate in contact with quartz				
11	0.00	0.01	15	15	0.00

[a] The difference between the amount of arsenic found on the sized crystalline zinc sulphide in this experiment and the amount of arsenate removed from solution by the powdered zinc sulphide specimens shown in Table 1 is due to the difference in the surface exposed to the solutions by the two varieties, since the former was minus 150-mesh while the latter was minus 200-mesh and powdery.

The Counterbalancing Ion

When a stable suspension of zinc sulphide in sodium arsenate solution is placed between charged electrodes, the particles migrate to the anode. The positive ions in the system are zinc and sodium.

The apparatus devised to determine which of these was the counter ion was an electrolytic cell comprising two platinum electrodes, each supported centrally in cylindrical nitrocellulose-membrane sacs about 2 cm. in diameter by 12 to 15 cm. long; the enclosed electrodes spaced 5 to 6 cm. apart in a Pyrex beaker. The space inside the membranes was partly filled with 0.002-molar sodium arsenate solution. The cell proper was filled as indicated with a stable suspension of zinc sulphide in 0.002-molar sodium arsenate solution. The electrodes were connected to a source of direct current and the electrolysis was continued for 20 to 40 min. The voltage drop across the electrodes was 25 volts or more.

The suspended zinc sulphide particles migrated toward the positive electrode and plated out on the nitrocellulose membrane in the form of a thick, heavy paste. The sulphate-ion and arsenate-ion concentrations increased in the annular space in the positive-electrode assembly. The liquid in the cathode space became alkaline to phenolphthalein, but gave negative tests for zinc as long as it remained alkaline. The experiment was repeated, adding 0.1 ml. of glacial acetic acid to 10 ml. of 0.002-molar sodium arsenate in the cathode space. The pH of this solution was 2.8. After 20 min. the cathode liquid gave a positive test for zinc ion and zinc plated out on the platinum as a fine, dark powder.

No migration occurred when the cell filling was zinc sulphide in distilled water.

Stability of Suspensions of Zinc Sulphide Particles in Aqueous Sodium Arsenate Solutions

Experiment IV.—A series of clean test tubes, each containing 20 ml. of sodium arsenate solution of the concentrations shown in Table 4 and one containing distilled water only, was

set up and to each was added one gram of minus 200-mesh zinc sulphide, after which the tubes were closed with clean paraffin-coated stoppers. Each tube was next shaken vigorously for 2 min. and allowed to stand undisturbed for 5 min. A clean pipette was dipped ½ in. under the surface of the liquid in each tube and a sm ll sample was withdrawn for microscopic examination. The suspensions in the test tubes were then allowed to stand undisturbed.

Microscopic examination of the preparations showed that the particles in the molar and 0.1-molar solutions were flocculated into aggregates of large size. Substantially all of the particles in the 0.01 and 0.001-molar preparations were in active Brownian movement and were unflocculated. Both flocculation and some sluggish Brownian movement were observed in the 10^{-4}-molar solution. The particles in 10^{-5} and 10^{-6}-molar solutions and in distilled water showed no Brownian movement and were highly flocculated.

The particles in 1.0, 0.1, 10^{-5} and 10^{-6}-molar arsenate solutions and in distilled water settled 5 in. in 30 min. or less, leaving a clear, supernatant liquid. The suspension in 10^{-4}-molar solution settled in 6 to 8 hr. That in the 0.01 and 0.001-molar solutions formed a stable suspension that did not settle in several days.

Mixtures (100 ml.) of the same liquid compositions as those in the test tubes, but with liquid-solid ratios of 10:1, were shaken to effect dispersion and then settled or centrifuged until clear liquid could be separated. The liquid samples then drawn were analyzed for zinc, with the results given in Table 4.

TABLE 4.—*Results of Analyses in Experiment* IV

Run No.	Concentration Na_2HAsO_4, Mols per Liter	Zinc Ion, Mg. per Liter
1	1.0	2.5-3.5
2	0.1	1.5
3	10^{-2}	<0.5
4	10^{-3}	<0.5
5	10^{-4}	3.0
6	10^{-5}	7.5
7	10^{-6}	8.0
8	0	>10

Experiment V.—A series of test tubes was set up, each containing 20 ml. of a sodium arsenate solution, the concentrations being as shown in Table 5. A solution containing 326 mg. of zinc ion per liter was added (stepwise) to each test tube until a precipitate was formed. Results are shown in Table 5. The precipitates formed in 0.01 and 0.001 molar arsenate solutions were heavier than those formed at other concentrations. The test-tube contents were separately filtered through asbestos in Gooch crucibles and the filtrates were tested for zinc. The filtrates from the molar and 10^{-4} and 10^{-5}-molar solutions showed about equal quantities of zinc, that from the 0.1-molar solution showed less and those from the 0.01 and 0.001-molar solutions showed substantially no zinc.

TABLE 5.—*Results of Experiment* V

Zinc Sulphate Solution Added, Ml.	Molar Concentration of Sodium Arsenate Solutions					
	1	0.1	0.01	10^{-3}	10^{-4}	10^{-5}
0.1	00	00[a]	00	00	00	00
0.3	00	sl. ppt.	ppt.	ppt.	00	00
0.5	00	ppt.	ppt.	ppt.	00	00
1.0	00	ppt.	ppt.	ppt.	00	00

[a] 00 = no change; ppt. = precipitate.

Precipitation of Stable Suspensions

Experiment VI.—The apparatus used was a cell such as has already been described, except that the anode was surrounded by two concentric nitrocellulose-membrane cells, the outer about 1 cm. greater in diameter than the inner; the axes substantially coincident. The cathode was surrounded by one sac as before. The annular anode space was filled with a stable suspension of zinc sulphide in 0.002-molar sodium arsenate solution while the inner anode space and the cathode cell were filled with 0.002-molar sodium arsenate solution. The cell filling was a stable zinc sulphide suspension in 0.002-molar sodium arsenate solution. Current was allowed to flow for 20 min. with about 25 volts across the electrodes.

The zinc sulphide in the anode annulus precipitated, leaving clear supernatant liquid. This precipitated material was removed, shaken with water, allowed to settle, and the water decanted. This operation was repeated several times. A portion of the exterior stable suspension was centrifuged to precipitate the solid, which was then washed with distilled water as the precipitated annular solid had been. Both

washed samples were analyzed for arsenic with the results shown in Table 6.

TABLE 6.—*Analyses for Experiment* VI

Run No.	Arsenic on 50-mg. Sample of ZnS from Stable Suspension, Mg.	Arsenic on 50-mg. Sample of ZnS Precipitated by Electrolysis, Mg.
1	0.200	0.040
2	0.200	0.050

DISCUSSION OF RESULTS

Experiment I demonstrates ion exchange between sodium arsenate in solution and the surface of zinc sulphide particles. It shows that the surfaces of such particles, even after thorough washing, yet hold sulphate and lower-sulphoxy ions although the wash waters are far below their saturation contents of the corresponding zinc salts. This behavior parallels that of galena with potassium ethyl xanthate, reported by Taylor and Knoll,[6] wherein also the rapid surface oxidation of natural heavy-metal sulphides in air and in water containing dissolved air was similarly established. Knoll[5] also showed that in the galena-xanthate system from 9 to 39 per cent of the xanthate ion abstracted was balanced by carbonate ion, from which it is assumed herein that the arsenate ion unaccounted for in Table 1 is similarly balanced. The possibility that the additional oxygen-carrying ions displaced by arsenate above those extractable by water (see Table 2) are due to oxidation by arsenate is excluded by reason of the parallelism of the phenomenon with that of such additional displacement by xanthate ion from galena, where the displacing ion is of reducing character.[6]

Experiment I demonstrates also that sodium ion takes no part in the negative-ion exchange, since its concentration is unchanged, within the limits of experimental error.

The indications of Experiment I are, therefore, that zinc sulphide particles in water consist at the particle surface of zinc ions and a mixture of oxygen-carrying ions comprising $SO_4^=$, $S_nO_m^=$, and $CO_3^=$ where m/n is less than 4, and that the negative ions will exchange with other negative ions in the water, if the latter form with zinc a less soluble salt. It is inferable that the completeness of the exchange increases with the degree of insolubility of the resultant salt.

Presence of the abstracted arsenate at the surfaces of the zinc sulphide particles is indicated by Experiment III, taken in conjunction with Experiment I. Experiment III was run with the crystalline material (lot 3) in order that the subsequent elutriation would wash out any zinc arsenate precipitated in the body of the solutions, such behavior having been proved by independent tests. Table 3 shows that 15 min. contact time is sufficient for equilibrium to be reached in 0.01-molar solutions (runs 3, 5, 6) and, therefore, that the equilibrium concentration at the surface is dependent upon the concentration in the solution (runs 1, 2, 3). Also, the extent of abstraction in a given time is proportional to the concentration in solution (runs 3, 7, 8, 9). Run 4 must, in the light of runs 3, 5 and 6, be disregarded.

Runs 10 and 11, Table 3, taken with the other runs recorded in the table, show that the arsenate abstraction is not simply a matter either of adsorption at any solid-liquid interface in contact with the solution (run 10), nor one of flocculation (run 11).

The experiments thus far discussed indicate that immersion of zinc sulphide particles in sodium arsenate solutions results in coating of these particles with zinc arsenate. Hypothesis that the zinc and arsenate in the surfaces analyzed are present in other than stoichiometric proportions must be excluded in light of the fact that the analyses followed 15-min. washings in running water. The possibility of excess of arsenate ion at surfaces im-

mersed in some of the more concentrated solutions is not, however, excluded.

The migration experiments (p. 495) showed that the sulphide particles were charged negatively in 0.002-molar arsenate solution, whence it is concluded that the anchored surface ion was arsenate. Increase in arsenate-ion concentration in the anode cell, taken with the appearance of sulphate ion therein, is interpreted to indicate release of arsenate ion by the charged particles reaching the anode membrane. This interpretation is supported by the experiments in the apparatus with two sacs around the anode, showing marked reduction in arsenate content of the discharged material (Table 6). The sulphate ion that accumulates within the inner anode membrane is taken to represent a part of the product of further oxidation of the discharged zinc sulphide, following the withdrawal of the arsenate coating. The presence of zinc in the cathode sac is thought to demonstrate that zinc was the ion counterbalancing the arsenate in the particle coating. Its failure to show in the tests wherein the liquor within the cathode membrane was alkaline is thought to have been due to the formation of a zincate. It cannot have been the result of reduction of zinc sulphide, because the membrane kept zinc sulphide particles too far away from the electrodes for such reaction to occur. The hydrogen-ion concentration was not sufficiently high to dissolve zinc sulphide.* Hence it is indicated that the zinc that plated out was that equivalently freed when the zinc sulphide-arsenate macro-ions plated out on the anode membrane; in other words, that the positive ion neutralizing the zinc sulphide-arsenate macro-ion was zinc. Additional evidence in support of this conclusion is found in Table 1, which shows that sodium ion is not removed from solution with arsenate ion and is, consequently, not bound with it by interionic forces.

Coagulation of the discharged particles in the cell and failure of the sulphide to migrate in distilled water is attributed to lack of a slightly soluble ionizable surface on the particles. Taggart[9] has shown that mineral particles disperse in aqueous solutions whenever and only when there are present in the solutions ions that can react with surface ions of the mineral to form a salt of limited solubility in the solution. In the present experiments the pressure of the electric field tore the arsenate ion away from the particles dammed by the anode membrane, while the sulphide in distilled water was coated principally with the relatively soluble sulphate and sulphoxy compounds of zinc.

The suspension experiments (series IV) demonstrate that permanent suspension occurs only when the surface coating is of very limited solubility. (The diethyl-aniline test for zinc ion employed herein is sensitive to 0.5 mg. per liter.) This condition for the present system corresponds to sodium arsenate concentrations of 0.01 to 0.001-molar (runs 3 and 4, Table 4). Zinc arsenate dissolves in solutions of higher concentration of sodium arsenate; in the solutions of lower concentration there is too little arsenate ion present to displace the oxidized sulphur ions and thus to anchor zinc ion in the immediately surrounding atmosphere. The runs of Experiment V, summarized in Table 5, are confirmatory, in that they show maximum precipitation of zinc arsenate and minimum concentrations of zinc remaining in solution at the arsenate concentrations corresponding to permanent suspension in Experiment IV.

Maximum Brownian movement and maximum suspension in Experiment IV correspond and correlate with maximum precipitation of zinc arsenate and minimum solution of zinc. The tests of Experiments IV and V, taken with the facts of Experi-

* Fales and Kenny[7] state that the optimum concentration of hydrogen ion for precipitation of zinc as the sulphide is from pH 2 to 3.

ments I to III, inclusive, and VI, indicate that maximum Brownian movement occurs when the sulphide-particle surface is most completely covered by ionized zinc arsenate, and that any serious diminution in such covering causes Brownian movement to lag or cease and flocculation to occur concurrently. The migration experiments give support to the picture that the zinc sulphide particles, when in Brownian movement, comprise macro-ions with a solid core of zinc sulphide and an adherent sheath of arsenate ions, and that these macro-ions hold around them a swarm of zinc ions. When the macro-ions are separated from the solution they carry the zinc ions with them (Table 4, runs 3 and 4). While the macro-ions are in suspension they are subject to the pulls exerted upon them by their surrounding swarms of zinc ions. These, in turn, are moved at random by their own interrepulsions and by the thermal movements of the water molecules. This results in ever-changing configurations of the zinc-ion swarm, with corresponding changes in the forces exerted thereby on the macro-ions. These respond by the erratic movements of Brownian motion.

Flocculation likewise is demonstrated to correlate with analytically measurable surface solubility. Under such circumstances the sulphide particles are no longer of macro-ionic character. Similar stoppage of Brownian movement and flocculation occur with lead and copper sulphides in the presence of xanthate ion,[1] which forms the corresponding xanthates of very low solubility. This is interpreted to mean that suppression of ionization of the surface similarly destroys the macro-ionic nature thereof.

It would seem to follow from the facts herein established, and from those set forth in references 1, 2 and 9, that whenever the finer fragments of a particular mineral species are dispersed in a flotation pulp, the surfaces of all of the particles of that species in that pulp are ionized, which is to say, water-wet, and cannot float. It does not, of course, follow that the non-dispersed particles will float, even though they may be flocculated, since, although flocculation is a concomitant of collector coating, it also occurs when particle surfaces are not collector-coaten; usually, in such circumstances, indicating surface solubility too great to anchor dispersing ions. It is to be further noted that the formation of a dispersing coating may be a useful intermediate step in collector coating. Thus sodium silicate disperses calcite by anchoring silicate ion and the calcite thereupon responds to a cationic collector; the calcite may be activated for sulphydrate collection by anchoring lead or copper ion; in both cases, the preliminary ion-anchorage step is indicated by dispersion of the fine calcite and the final collector-coating by flocculation thereof.

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Collector Coatings in Soap Flotation

By Arthur F. Taggart,* Member, and Nathaniel Arbiter,† Junior Member, A.I.M.E.

(New York Meeting, February 1943)

The fact that the floatability of minerals with fatty-acid collectors changes as the pH of a pulp varies was utilized in the early days of flotation, when sulphuric acid was used with oleic acid to float sphalerite and galena from gangues which themselves float with fatty acids in alkaline pulps. Gaudin[1] showed that this difference in floatability is a regular function of the extent of the change in pH. The present paper offers an explanation of the phenomenon, and on the basis thereof sets up recommendations for control of soap-flotation operations.

Experimental

Parallelism between floatability of a mineral in a given environment and its contact angle in that environment is so well established,[2] and the contact angle is so readily and rapidly measurable, that contact angles have been used throughout the experimental work to indicate relative floatability.

Experiments comprised contact-angle measurements on a number of representative minerals and metals, and a variety of subsidiary tests designed to check and to amplify the contact-angle indications.

Contact-angle measurements were made by the method described in detail by del Giudice.[3] Sequence of procedure for determination of the points of Fig. 1 was:

1. Polish specimen. Most specimens yielded zero contact angle after polishing; a few showed a slight "cling." Cling is characteristic of specimens that pit on polishing; it is due to a residue of organic contamination, originally on the particle surface, which persists in the pits. The cling did not change total contact angle after conditioning, and is not, therefore, additive in the upper part of the curves of Fig. 1. It may have broadened the bases of the curves for the affected minerals slightly.

2. The specimen was next conditioned in soap solution for 5 minutes.

3. The conditioned particle was transferred, under conditions that precluded surface drying, to an aqueous solution of the pH desired and contact angle determined.

4. The particle was thereupon transferred, again without surface drying, to water.

The confirmatory tests will be described at the places pertinent in the argument.

Discussion of Results

Gaudin and Vincent[4] have shown that contact angles with heptoic acid as a collector decrease for a given specimen when the concentration of collector present falls below a certain value. A similar reduction in angle occurs with collector-conditioned particles that are visibly slime-coated, and varies roughly with the extent of coating. Since in the last case the reduction in angle is certainly due to reduction in the proportion of total conditioned mineral surface presented to the bubble, the authors believe that the same condition prevails in the first case cited, and that similar reduction in angle and floatability will occur whenever starvation quantities of collector are present, whether by design or by reason of excessive consumption.

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[1] References are at the end of the paper.

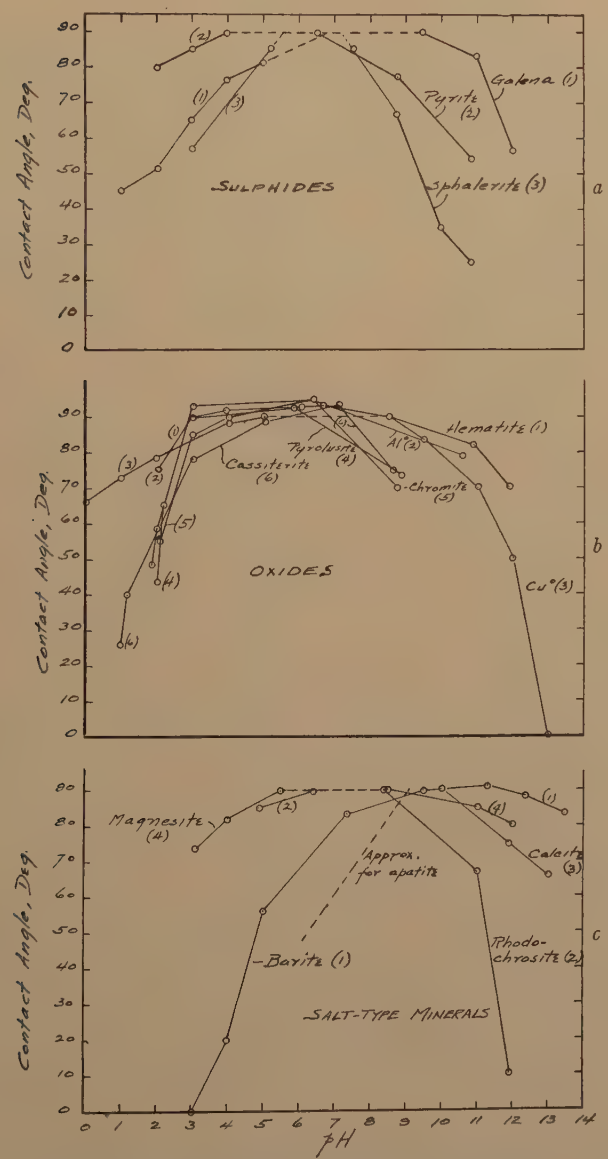

FIG. 1.—CONTACT ANGLES IN AQUEOUS SOLUTIONS.

Points in the pH range 5 to 9 were obtained in most cases by the use of a borate-boric acid buffer; those on the acid side, by sulphuric acid; those on the alkaline side by sodium hydroxide.

All points are averages of several determinations, which showed, in most cases, not over 5° spread between maximum and minimum angles.

Angles on chromite on the alkaline side dropped from 10° to 20° on standing in the test solution.

It has been shown[5] that when a collector that forms a coating on the mineral of relatively high solubility is used alone, excess of collector must be maintained in the

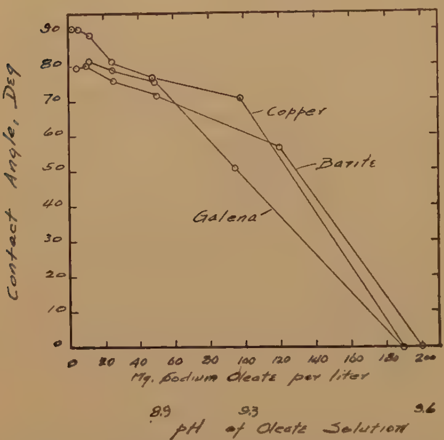

FIG. 2.—CONTACT ANGLES IN SOAP SOLUTION.

solution in order to maintain a contact angle; further, the magnitude of the angle and the rapidity with which it is attained increase with increase in collector concentration. Contact is also improved with such collectors by adding to the solution the ion contributed by the mineral to the coating compound; conversely, contact is hindered by addition of the ion displaced by the collector ion. Thus laurylamine hydrochloride is a collector for barite and the amine ion is abstracted in the collecting reaction; addition of sulphate ion (H_2SO_4; Na_2SO_4) increases the contact angle with a given concentration of the amine salt, or causes a given contact angle to be attained with a lower concentration of amine salt; barium ion ($BaCl_2$) hinders the coating. All of these amine behaviors indicate that the collector reaction is controlled by a solubility-equilibrium relationship, and that the low contact angles with the amine collectors correspond to partial surface coatings.

If a fractional contact angle reflects fractional collector coating, the tests summarized in Fig. 1 indicate that the effect of acid and alkali on soap-type collector coatings is to control the number of outwardly oriented hydrocarbon groups at the particle surface.

It was found that, in general, a soap-conditioned particle that had shown a low contact angle in acid or alkaline solution would, when transferred thence to distilled water, with no intervening treatment, exhibit a higher angle. The copper specimen, after such testing, was transferred next to an aqueous solution of pH corresponding to its maximum contact angle in soap solution (Fig. 2). It again attained that angle. These tests show that the soap coating persists and that the fatty-acid ion is present at the surface throughout the periods of depression in acid and alkali. Sphalerite was an exception, in that depressed contact angles did not recover on removal from the depressing solutions.

Attainment of the equilibrium contact angles on transfer to new solution environments required time. This indicates that some sort of equilibrium change was taking place. That it was not removal of the surface coating has just been shown. It is thought to have involved the formation of so-called acid or basic soaps at the surface, according to the pH. The supporting evidence follows.

Langmuir and Schaeffer[6] have shown that neutral alkaline-earth soaps are transformed in part to acid soaps with increase in hydrogen-ion content of the environment. Their work further shows that suitable solvents remove fatty acid from such altered soap, leaving voids of molecular size in the leached films. Hence the present authors believe that, to the left of the optima of Fig. 1, oleic acid molecules, in quantities proportionate to the concentration of hydrogen ion in the solution, are present in the soap layer and that they are held there by attraction between the

oleate radicals of the acid and the surface soap. Such molecules should orient with their carboxyl ends outward. This would both decrease the percentage of hydrocarbon surface presented to the water and substitute water-avid for water-repellent surface to the extent of the conversion.

Both Soyenkoff[7] and McBain[8] report basic soaps formed by addition of hydroxyl ion. As far as data are available, the alkaline branch of the curves in Fig. 1, for each of the minerals and metals tested falls in the pH range in which the corresponding metallic hydroxides precipitate. Hence there is in the case of each a hydroxyl-oleate contest for the anchored metal ion. That this should result in progressive displacement of oleate ion as the hydroxyl-ion concentration of the environment increases is completely in accord with the kinetic concept of equilibrium. But again the displaced oleate must remain at the surface to satisfy the condition for return of full angle in aqueous solution at optimum pH. It is postulated that the displaced oleate ions are held approximately in place by mutual attraction for the yet bound oleate ions of the metal-soap coating, but are free to orient with carboxyl end toward the water. In this case the counter ion would be the cation with which hydroxyl ion was introduced. The observed reduction in contact angle with increasing pH is thus attributed to progressive reduction in the percentage of hydrocarbon groups outwardly oriented, and to the water-avid character of the metal hydroxide and the carboxyl.

The reason for the discordant behavior of sphalerite is unknown.

The progressive drop in contact angle that occurs in solutions of high soap concentration (Fig. 2) reflects common operating experience. The following argument goes to the probable cause. Studies of multilayered metallic-soap films[9] and of the crystal structure of the metallic soaps[10] shows that both comprise successive parallel layers of the compound alternately oriented hydrocarbon-to-hydrocarbon and carboxyl-to-carboxyl. When a mineral particle completely depressed in a high-soap solution is brought through the surface of the solution into air, it immediately sheds solution and dries. It exhibits full contact angle when reimmersed in aqueous solution of optimum pH (Fig. 1). These facts are consistent with the postulate that in the high-soap solution a complete metallic-soap film with hydrocarbon outwardly oriented exists directly at the mineral-particle surface, and that a more or less complete second film of molecular metallic soap (or, possibly, alkali-metal soap or oleic acid), oppositely oriented, adheres thereto. Multilayers of collector compounds at mineral surfaces are well known. Precipitation of alkali-metal soap or oleic acid would simply constitute redistribution of the micellar material, which comprises these substances[11] and is always present in soap "solutions" of high concentration. Such redistribution would follow from the tendency for large bodies of precipitate to grow in preference to small.

The function of oil in soap flotation is clearly shown by contact-angle tests between Nujol, a heavy petroleum oil, and barite in a soap solution containing 200 mg. per liter of sodium oleate. The air-bubble angle in this solution is zero. The oil angle is 165°. It is apparent that the liquid hydrocarbon can displace the second layer of soap with great ease. When an air bubble is now brought into contact with the particle at a considerable distance from the residual oil droplet it makes no contact. When it is slid along the surface toward the droplet, it suddenly makes contact with an angle of 75° to 80°, while it is yet at a considerable distance from the visible oil. If the bubble is brought into contact with the oil droplet, oil flashes around the bubble and "cements" the particle to it, but the bubble will not now make contact with the particle through the oil layer.

When an oil droplet is brought into contact with a soap-conditioned barite particle in water at pH 4 (contact angle with air equals 20°), the oil droplet spreads to an angle of about 45°. This oil will spread on an air bubble brought into contact with it and thus attach the particle to the bubble. Thus the use of oil with soap permits wider latitude in soap concentration and pH than would otherwise be possible, but, conversely, it decreases the sharpness of control possible by variation in pH.

An indication of the possibilities of prediction from the data presented herein follows: Sphalerite should be activated by oleate ion and excess oleic acid at a pH between 1 and 2 (Fig. 1*a*), on the basis of the barite-oiling test cited on p. 503. Malachite would not be expected to activate at this pH, despite the copper-metal curve on Fig. 1*b*, because the reaction between acid (sulphuric) and carbonate tends to loosen any surface layer of copper oleate that might form. Malachite should, however, activate on the acid side near pH 7. Tricalcium phosphate should not activate to an effective extent until close to pH 9, since the acid branch of calcium-salt minerals falls to the right of that for the barium minerals (Fig. 1*c*). Quartz is not activated by soap.

An artificial mixture of sphalerite, malachite, and pebble-phosphate debris was deslimed, acidified until strongly red to litmus and then floated with oleic acid, yielding a good sphalerite concentrate. Sodium hydroxide was then added to the pulp in the machine until litmus was only slightly reddened, when much of the malachite floated. Further sodium hydroxide was added until the litmus turned pink, whereupon part of the phosphate floated. The remainder was brought up readily by addition of a little fuel oil. Quartz was the final tailing. Separation was good, judged by microscopic examination.

The fact that this test preceded by several years the explanation of its cause does not detract from its confirmatory character.

Summary

The experimental data indicate that:

1. The optimum pH for soap flotation of any mineral is close to that pH at which the hydroxide of the corresponding metal precipitates.

2. Maximum contact angle indicates complete surface filming with a hydrocarbon-bearing compound oriented hydrocarbon-end outward.

3. Contact angles less than the maximum indicate fractional hydrocarbon-like surfaces.

4. Metal-soap surface films are resistant to removal by acid and by alkali over a wide pH range.

5. The contact angles of metal-soap collector films and, consequently, the floatability of minerals so coated are controllable through a correspondingly wide pH range.

6. The mechanism of surface change is probably partial conversion to acid or to basic metallic soap at the particle surface; this is a reversible reaction.

7. The change in chemical character of the surface compound is probably accompanied by a change in orientation of a corresponding part of the hydrocarbon-bearing groups in the surface layer.

8. Acid or basic metallic soaps precipitate as such at mineral surfaces in equilibrium with neutral soaps, in the same proportions as those to which a neutral soap coating is converted at the same pH. Subsequent adjustment of the pH to a point near the optimum will be necessary, however, for effective flotation without oil. This permits conditioning to be effected at a pH greater than 7, in most cases, where dispersion of the compound bearing the soap ion is easy.

9. Collector-coating reactions involve mass-action equilibria with smaller con-

stants than prevail for normal precipitation of the corresponding compounds.

10. The depressing effect of an excess of alkali soap in soap flotation is due to the deposition of a more or less complete second layer of soap and/or soap acid, oriented reversely to the original layer. Depression may be corrected by use of a mineral oil.

11. Strong flotation may be expected to occur throughout the region of contact angles greater than 60°.

12. Differential soap flotation of certain of the minerals of Fig. 1 from others is possible by adjustment of pH.

13. It is not necessary for a mineral to free detectable amounts of its characteristic metallic ion in the flotation pulp for that mineral to be amenable to soap flotation. Oxides such as Fe_2O_3, Fe_3O_4, SnO_2 and Al_2O_3, which do not free metal ions in water to an appreciable extent, will yet react with alkali-metal soaps and become coated with the corresponding metallic soaps.

14. Optimum concentration of soap for conditioning lies in the range of 20 to 50 mg. per liter of water. A deficit may be brought about, despite such an initial addition, if the pulp contains either a considerable quantity of metallic ions capable of precipitating insoluble soap at the pH prevailing, or a considerable quantity of a fine metallic oxide capable of reacting with the soap.

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DISCUSSION

(*S. J. Swainson presiding*)

G. Gutzeit,* Westport, Conn.—This excellent paper brings systematic experimental proof as well as a theoretical interpretation of the fact (already known to flotation experimenters and indicated by different authors) that the pH is one of the main controlling factors in the soap flotation of minerals; i.e., that a given species of mineral will show a tendency to float within a certain typical pH range, when fatty acids, their derivatives, or their salts are used as collectors. Differential flotation of oxide minerals (ilmenite, monazite, zircon, rutile) with fatty-acid sulphates, based upon close control of the pH, has been applied by the writer and P. Kovaliv to the Egyptian "black sands," in 1939.[12]

It is shown by Taggart and Arbiter that, once the mineral filmed with the corresponding metal soap, the coating will persist despite wide changes in pH. The formation of acid and alkaline soaps is postulated, and furthermore, the freedom for part of the oleate ions to reverse their polar orientation. This is in accordance with the experiments of N. K. Adams and Miller (University College, Southampton, England), who found that there is a very large diminution in the lateral adhesion between the molecules of a fatty-acid film on water solutions, when the end groups are ionized by a change in acidity.

It is interesting to note that (with the odd exception of magnesium) the highest contact angles obtained by the authors of this paper on any given mineral containing a certain metal is roughly in the same pH range as the optimum flotation conditions for quartz after activation by the *same* cation. Table 1 will substantiate this correlation.

In the summary, there are two statements (Nos. 1 and 13) which, amazingly for a paper from the School of Mines, Columbia University,

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* Westport Mill Research Division, The Dorr Company.

[12] G. Gutzeit and P. Kovaliv: *Archiv. Sc. Phys. et Nat.* Geneve, (1939) **21**, 260.

seem to support the so-called "adsorption theory" of flotation, although the way they are expressed tends to suggest the contrary. The first one ("Collector coating reactions involve mass-action equilibria with smaller constants than prevail for normal precipitation of the corresponding compounds") is not sufficiently substantiated by experimental evidence, but certainly involves a departure from the "chemical theory." The second one ("It is not necessary for a mineral to free detectable amounts of its characteristic metallic ion in the flotation pulp for that mineral to be amenable to soap flotation . . . ") is corrected by the word "detectable" and by the statement that it remained nearly constant until pH = 1. This is due to the fact (as pointed out by Frumkin) that, when ionization takes place, the dipole moment at the end of the molecule is very much altered by the development of a pair of new electric charges. The molecules are oriented in the films with the positive carbon of the carboxyl above the negative oxygens, and if the hydroxyl group is not dissociated, the resultant dipole is large, with the positive end uppermost. If the hydroxyl is ionized, there is an additional dipole consisting of the positive ion below and the negative oxygen above; this more than neutralizes the dipole present in the fatty acid.

TABLE 1

Contact Angle 90° or Above[a]		Quartz Flotation[b]			pH of Precipitation of the Hydroxide[c]	
Mineral	pH Range	Activated Salt	Recovery 80 Per Cent	Optimum Recovery	Ion	pH
Galena	6.5– 9.3	$PbCl_2$	5.0– 9.5	6.0– 9.2	$Pb^{\cdot\cdot}$	6.0
Pyrite	4.0– 6.5	$FeCl_3$	2.0– 8.2	3.5– 7.2	$Fe^{\cdot\cdot\cdot}$	2.0 (?)
Sphalerite	5.5– 7.1	$ZnSO_4$	6.5– 7.8	6.8– 8.5	$Zn^{\cdot\cdot}$	5.2
Barite	9.5–12.0	$BaCl_2$	9.0–14.0	9.5–14.0	$Ba^{\cdot\cdot}$	13[d]
Calcite	10.0	$CaCl_2$	9.0–14.0	9.5–14.0	$Ca^{\cdot\cdot}$	12[d]
Hematite	3.0– 8.0	$FeCl_3$	2.0– 8.2	3.5– 7.2	$Fe^{\cdot\cdot\cdot}$	2.0
Copper	5.0– 8.5	$CuSO_4$ or $CuCl_2$	5.8– 9.5	6.5– 9.0	$Cu^{\cdot\cdot}$	5.3
(Magnesite	5.5– 8.5	$MgCl_2$	8.0–12.0	9.5–11.5	$Mg^{\cdot\cdot}$	10.5)

[a] Taggart and Arbiter: page 500, this volume.
[b] Kraeber and Boppel: *Metall. und Erz* (1934) Haft 19, **31**, 421. C. E. Mosmann: Doctor's Thesis, Univ. of Geneva (Switzerland), 1942.
[c] H. T. S. Britton: Hydrogen Ions. London, 1929. Chapman and Hall, Ltd.
[d] The carbonates of calcium and barium already precipitate at a lower pH.

nevertheless, these minerals will become coated with the corresponding metallic soaps. Although this is the probable mechanism, it should be kept in mind that another explanation for the attachment of a fatty-acid molecule on a mineral surface (as well as the variation of the contact angle according to the degree of ionization of the surface-anchored hydrocarbon compound) is still possible.

Following Frumkin[13] confirmed by Schulman and Hughes,[14] films of free fatty acids give positive surface potentials, while those of their salts show small negative potentials. Thus, a film of myristic acid on water (constant minimum area of 20 sq. A) had a negative potential of minus 50 mv., at pH 12, while, between 11 and 4 it rose rather steeply to 400 mv., where

[13] Frumkin: *Ztsch. physikal. Chem.* (1924) 3, 194; (1925) 116, 494.
[14] Schulman and Hughes: *Proc.* Roy Soc. (1932) **A-138**, 436.

Furthermore, in order to explain the depression observed in solutions of high soap concentrations, formation of a second reversely oriented soap (or fatty acid) film is postulated. Such a second layer is thought to adhere to the first one through van der Waals' forces between the hydrocarbon groups. Easy displacement of this second layer by a nonpolar hydrocarbon (Nujol) is also admitted. Therefore, pure adsorption phenomenons are used in order to explain secondary flotation reactions. But van der Waals' forces can also be active directly between a mineral surface and an organic compound. Collection of molybdenite by kerosene is certainly unexplainable by any "chemical theory."

The purpose of these remarks is not to back the adsorption theory of flotation, but only to show that no absolute argument can be opposed to either hypothesis. Moreover, there is no

absolute border between these different types of attachment. As Glockler[15] points out, the classification of forces between atoms as covalent, van der Waals, and others is arbitrary and artificial. "Although the magnitudes of van der Waals' energy and bond energy are usually of different orders, the quantum mechanical treatment does not necessarily recognize any sharp difference in the *nature* of these two types of attractive force. Both the physical type of cohesion and the chemical type are essentially electrostatic in origin. Indeed, it may be entirely possible for the van der Waals' attraction to be of the same order of magnitude as bond energies in the case of very large molecules. This view, of course, bears directly upon the long disputed question as to where, from the standpoint of combining forces, adsorption ceases and compound formation begins."[16]

[15] Glockler: *Jnl. Chem. Phys* (1939) 2, 823.

[16] C. S. Stillwell: Crystal Chemistry, 211. Ref. also p. 168, last paragraph. New York, 1938. McGraw-Hill Book Co.

Nature of the Adsorption of Fatty Acids from Organic Solvents by Inorganic Lead Compounds

By Alexander Knoll* and Dwight L. Baker†

(New York Meeting, February 1941)

The work herein reported shows that galena in certain organic solutions of fatty acids becomes coated with lead soaps, and that this coating is not only highly water-repellent but is also repellent to certain organic liquids, particularly those of high surface tension; i.e., nitrobenzene, diphenylmethane.

Wetting Properties of Galena in Nonaqueous Fatty Acid Solutions

When a "captive bubble"[1] is withdrawn from a polished, plane galena surface under a nitrobenzene solution of stearic acid, the air bubble shows a decided tendency to "cl ng" to the mineral surface. The contact angle, therefore, was greater than zero, although it was not large enough to be measured quantitatively by the captive-bubble method.* When pure nitrobenzene was substituted for the nitrobenzene-stearic acid solution, no clinging of the bubble was observed; viz., contact angle = zero.

Although the contact angle developed in air-galena-nitrobenzene-fatty acid solution was small, the organic solvent repellency of the galena surface was great enough to enable good recovery in a laboratory flotation cell.[2] In Table 1 are given the results of flotation experiments using several different fatty acids dissolved in nitrobenzene.

Table 1.—*Results of Flotation Experiments*

Concentration[a]	Percentage Recovery in Five Minutes								
	Stearic Acid, C_{18}	Magaric Acid, C_{17}	Palmitic Acid, C_{16}	Myristic Acid, C_{14}	Tridecylic Acid, C_{13}	Lauric Acid, C_{12}	Undecylic Acid, C_{11}	Capric Acid, C_{10}	Caproic Acid, C_6
None Blank—no acid	10	9	9						
0.0002 M		27	80	32	23	28	23	13	
0.0006 M		68	91[b]	83	64	65	55		
0.0014 M	95[b]	96[b]	97[b]	93[b]	90[b]	95[b]	88[b]		
0.003 M	97[b]	97[b]	98[b]	96[b]	93[b]	97[b]	95[b]	75	10

[a] 10 grams of galena and 50 c.c. of nitrobenzene solution of acid were added to the cell. During the run more solution (30 c.c.) was added dropwise to keep a steady overflow during the 5-min. run. The heavy froth (rapid recovery) in some cases (marked *b*) made the addition of solution unnecessary. Higher concentrations of acid gave a heavy froth equivalent to the galena-aqueous xanthate system.

The material reported in this paper is taken from a dissertation presented by Dwight Baker to the Faculty of Pure Science, Columbia University, in partial fulfillment of the requirements for the degree of Doctor of Philosophy. For details see original dissertation. Manuscript received at the office of the Institute Dec. 2, 1940. Issued in Mining Technology, May 1941.

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[1] References are at the end of the paper.

* It is difficult to obtain reproducible contact angles less than 40° by means of the captive-bubble method.

TABLE 2.—*Removal of Stearic Acid by Galena from Several Organic Solvents*

Solvent	Original Solution Stearic Acid, Mg. per 250 C.C.	Time of Shaking	Stearic Acid Removed, Mg. per 100 Grams Galena	
			By Titration	By Dry Weight
Nitrobenzene	375	10 min.	48.2	
Nitrobenzene	375	6¼ hr.	63.5	
Nitrobenzene	375	24¼ hr.	80.0	
Toluene	490	5 min.	45.2	25
Toluene	490	15 min.	36.2	35
Toluene	490	30 min.	46.0	50
Toluene	490	1 hr.	45.2	60
Toluene	490	3 hr.	55.0	55
Toluene	490	8 hr.	63.5	80
Toluene	490	24 hr.	90.5	88
Toluene	500	1 min.	38.5	
Toluene	500	1 hr.	50.5	
Toluene	500	7¾ hr.	70.3	
Toluene	500	24 hr.	101	
Toluene	500	48 hr.	151	
Toluene	500	73 hr.	184	
Dioxane	510	10 min.	45.2	43
Dioxane	510	1 hr.	50.5	50
Dioxane	510	6¾ hr.	51.7	73
Dioxane	510	24¼ hr.	67.0	83
Mixed heptanes	495	10 min.	46.2	40
Mixed heptanes	495	1 hr.	50.5	50
Mixed heptanes	495	5½ hr.	66.2	88
Mixed heptanes	495	23 hr.	114	117

REMOVAL OF FATTY ACIDS FROM NONAQUEOUS SOLUTIONS BY GALENA

When 20 grams of minus 200-mesh galena was mechanically shaken with 50 c.c. of several organic-solvent solutions of stearic acid, it was found that stearic acid was removed from solution by the galena particles. The results of these experiments are given in Table 2.

The removal of stearic acid was determined by (1) titration with 0.01 N aqueous sodium hydroxide using phenolphthalein as indicator, and (2) in some cases by evaporation of solvent under diminished pressure and subsequent weighing of the fatty acid residue. (Unless otherwise stated all removals were determined by the titration method.)

The removal of other fatty acids by minus 200-mesh galena was determined with toluene as a solvent medium (Table 3). The concentrations and removals in this table are given in millimols, in order that the removals may be compared on a molecular basis.

TABLE 3.—*Removal of Fatty Acids from Toluene by Galena*

Acid	Original Solution Concentration, Millimols per 250 C.C.	Time of Shaking	Removal of Acids, Millimols per 100 Grams Galena
Acetic	19.8	10 min.	1.75
Acetic	19.8	24¼ hr.	3.02
Capronic	16.4	10 min.	1.21
Capronic	16.4	6 hr.	1.75
Capronic	16.4	24 hr.	3.18
Heptyllic	16.1	10 min.	.77
Heptyllic	16.1	6 hr.	1.31
Heptyllic	16.1	22 hr.	1.98
Capric	16.8	10 min.	1.22
Capric	16.8	6 hr.	1.70
Capric	16.8	24¾ hr.	1.04
Undecylic	17.5	10 min.	1.02
Undecylic	17.5	6 hr.	1.33
Undecylic	17.5	24 hr.	3.12
Stearic	17.6	5 min.	1.59
Stearic	17.6	8 hr.	2.24
Stearic	17.6	24 hr.	3.19

REMOVAL OF STEARIC ACID BY ANGLESITE AND CERUSSITE

When minus 200-mesh anglesite and cerussite were shaken with a toluene or nitrobenzene solution of stearic acid, the fatty acid was adsorbed by the powdered minerals (Table 4).

The Adsorbed Film

Galena particles that had been shaken with nonaqueous solutions of fatty acids were separated from the solutions by filtration, washed repeatedly with excellent solvents for stearic acid (viz. petroleum ether, ethyl ether, benzene and alcohol) dried in air and finally brought into contact with (1) nitrobenzene and (2) water. In each case the galena particles treated thus were not "wet" (except by external work) by nitrobenzene and were highly water-repellent. Untreated galena particles were immediately "wet" by both water and nitrobenzene.

If washed galena particles that had been shaken with a toluene solution of stearic acid with cold organic solvents were further washed with a hot mixture of butyl alcohol and xylene (1:9), there was removed from the "treated" mineral surface a substance that precipitated out of the hot washings on cooling. The anslysis of a sample of this substance showed that it had a melting point of 111° to 112°C. (normal lead stearate has a melting point of 109°) and a lead content of 26.9 per cent (normal lead stearate is 26.7 per cent Pb).

The behavior of cerussite resembles that of galena. It collected in the air-liquid interface of the froth when shaken with a nitrobenzene solution of stearic acid. When separated from nonaqueous solutions of fatty acids, washed and dried, cerussite exposed a surface that was repellent to both water and nitrobenzene. A substance containing lead and resembling lead stearate in appearance and solubility characteristics was removed from the treated cerussite by the hot mixture of butyl alcohol and xylene.

Anglesite exhibited properties quite different from those of galena and cerussite. Although anglesite when shaken with a nitrobenzene solution of stearic acid collected in the extensive air-liquid interface of the froth, it did not keep its nitrobenzene-repellent properties when separated from the fatty acid solution and washed with organic solvents. The treated anglesite was also readily wet by water. When treated anglesite was extracted with the hot mixture of butyl alcohol and xylene, no substance resembling lead stearate could be detected in the washings.

Table 4.—*Removal of Stearic Acid by Anglesite and Cerussite from Organic Solvents*

Adsorbent	Solvent	Original Solution Concentrated Stearic Acid, Mg. per 250 C.C.	Time of Shaking	Stearic Acid Removed, Mg. per 100 Grams
Anglesite	Toluene	502	10 min.	45.0
Anglesite	Toluene	502	6½ hr.	62.7
Anglesite	Toluene	502	24¾ hr.	64.5
Cerussite	Toluene	502	10 min.	68.5
Cerussite	Toluene	502	6 hr.	67.5
Cerussite	Toluene	502	24 hr.	121
Anglesite	Nitrobenzene	288	10 min.	45.2
Anglesite	Nitrobenzene	288	6¼ hr.	55.0
Anglesite	Nitrobenzene	288	24 hr.	59.0
Cerussite	Nitrobenzene	288	10 min.	73.5
Cerussite	Nitrobenzene	288	6¼ hr.	89.2
Cerussite	Nitrobenzene	288	24¼ hr.	115

Discussion of Results

The fact that finely ground galena can be floated in a solution of fatty acid in nitrobenzene indicates that the galena under these conditions exposes a nitrobenzene-repellent surface. This nitro-

benzene-repellency of galena has been further demonstrated by means of the captive-bubble test.

It has been found that the change of wetting properties mentioned above is accompanied by the adsorption of fatty acid by the galena surface. That the adsorption is accompanied by a chemical reaction between fatty acid and the inorganic lead salts of the galena surface, resulting in the formation of a surface film of lead soaps, is suggested by an analysis of the experimental results.

In the first place, the surface change is of a permanent nature, persisting after the galena surface has been washed repeatedly with good fatty acid solvents. Furthermore, when sufficiently severe methods were used, it was possible to remove from the treated galena a substance that possessed the chemical and physical properties of a lead soap (lead stearate).

Additional evidence is supplied by the adsorption characteristics. In Tables 2 and 3, it is shown that the adsorption of fatty acids was a function of time of contact between the phases. Equilibrium was not reached apparently even after 73 hr. of shaking. Nonchemical adsorption usually reaches equilibrium very rapidly.[3] Further, one of the more valid generalities in the field of capillarity is that in nonchemical adsorption the adsorption is greater the less the solvency of the solvent.[4] Thus the independence of the adsorption of stearic acid with change in solvent (Table 2), although the solubility of stearic acid in these solvents varied greatly, is evidence that the adsorption is chemical. Finally, the approximate equivalence of the removal of the several fatty acids (Table 3) is a result that is consistent with the chemical hypothesis.

The results of experiments using anglesite and cerussite supply more detail to the picture of the adsorption mechanism. They suggest that the reaction between the fatty acid and the galena surface is not general. In particular, the behavior of cerussite suggests that one of the salts associated with the galena surface that may enter into chemical reaction with stearic acid is the lead carbonate.* The behavior of anglesite, on the other hand, indicates that the final oxidation product of the galena surface, lead sulphate, does not react with the stearic acid to form a lead soap.

Undoubtedly the reactions responsible for the change in wetting properties of galena after contact with solutions of fatty acids in nonaqueous solvents differ markedly in degree, if not in kind, from those taking place in analogous aqueous systems. It is hardly possible that common ion-exchange reactions are responsible, as the solvents used are regarded as nonionizing, and means were taken to work under rigidly anhydrous conditions. Kahlenberg[6] and Koenig[7] have demonstrated that instantaneous exchange reactions can take place in carefully dried solvents of low conductivity. Thus on mixing anhydrous solutions of hydrogen chloride and metallic soaps, an instantaneous precipitation of the metallic chloride occurred.

Also, exchange and redistribution reactions involving atomic groups are common in organic systems. As the work here reported was of an exploratory nature, no attempt was made to define rigidly the type of reaction responsible for the observed effects.

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* The anions carbonate, sulphate, sulphite and other reducing sulphoxides are associated with the surface of air-ground galena.[5]

DISCUSSION

(A. F. Taggart presiding)

O. A. Ziehl,* Elizabeth, N. J.—The recovery figures shown in Table 1 seem to point to a fairly persistent, though not very pronounced, general trend of not only decrease in effectiveness with decreasing molecular weight of the fatty acids but also of periodic variations within the series. The fatty acids having an odd number of carbon atoms, down to C_{11}, appear to be less effective than their neighbors having an even number of C atoms, especially at the lower concentrations. The effectiveness of the acids as collectors is in line with their melting points:

Number carbon atoms	18	17	16	14	13	12	11	10
Melting point, deg. C	70	62	63	54	41	44	28	31

Unless the differences in the results lie within the error of measurement and, therefore, are merely coincidental with the periodic ups and downs in the melting-point series, it is possible that the various acids may form soaps having solubilities in direct relation to the melting points of the respective acids.

That solubility of the resulting soap does play a part seems to be evident from the fact that oleic acid is a poor collector in organic liquids, lead oleate being far more soluble in an organic liquid (ether) than the lead salts of stearic and palmitic acids.

The authors cite proof of formation of lead stearate by the interaction of galena and of cerussite with stearic acid dissolved in an organic liquid. This reaction implies the displacement of the negative radical with which the lead was combined originally. It would be of interest to know what happens to this inorganic radical. The behavior of anglesite also could bear closer scrutiny.

* Chemist, Metals Disintegrating Company.

R. R. Arellano,* Cambridge, Mass.—I wonder whether any difference has been noticed in the behavior of saturated and unsaturated fatty acids. In the flotation of manganese ores the unsaturated sodium soaps are much better collectors than the saturated ones.

* Massachusetts Institute of Technology.

Organic Sulphides as Oily Collectors

By M. D. Hassialis,* Member A.I.M.E.

(New York Meeting, February 1943)

The claim is made in a number of patents[1,2,3,4] that some compounds of the class known as aryl sulphides have collector properties. One of these patents generalizes the claim to include all aryl sulphides, whose generic formula is $R\text{-}S_x\text{-}R'$ (where $x = 1,2, \ldots$), with the restriction that if the aromatic radicals R, R' contain substituent groups these groups may not contain nitrogen or oxygen. Moses, Hess, and Perkins cite as evidence for their claims the results of flotation tests on Anaconda and Miami copper ores using diphenyl sulphide, ditolyl disulphide and dinapthyl disulphide as collectors. The use of alkyl sulphides as collectors has also been reported[5] with the conclusion that the alkyl sulphides are better collectors than the corresponding aryl sulphides. The experimental work cited appears to confirm this contention.

In order to test these claims with respect to diphenyl sulphide, a sample, purchased from the Eastman Kodak Co., was tested against Anaconda ore. The procedure followed was that of Moses, Hess, and Perkins. A sample of ore weighing 500 grams was placed in a laboratory ball mill. To this was added 220 grams of water, 0.06 grams of commercial diphenyl sulphide (¼ lb. per ton of ore), and 0.8 grams of lime (3 lb. per ton of ore). The material was ground until 93 per cent passed a 65-mesh screen. This pulp was diluted to 16 per cent solids. A sample of the diluted pulp was placed in a Minerals Separation type of laboratory flotation cell. After a 2-min. agitation period, pine oil was added and the froth collected for a period of 15 min. This test recovered 68 per cent of the copper, with a ratio of concentration of 6.8:1. The tailing assay was 0.3 per cent Cu. Moses, Hess and Perkins attained a recovery of 75 per cent, with a ratio concentration of 12:1. Although these results do not compare too well with those of Moses, Hess and Perkins, subsequent tests showed an improvement, as judged by vanning of the tails. It is concluded, therefore, that the patent results are confirmed.

Contact-angle Tests

In order to study the response of the various minerals, found in Anaconda ore, to commercial diphenyl sulphide, some contact-angle tests were made. Polished particles of bornite, chalcocite, chalcopyrite, pyrite and quartz were conditioned for 15 min. in a suspension of diphenyl sulphide in water. The concentration of this suspension was 0.284 grams per liter, this being the concentration of this reagent in the ball mill. The conditioned particles were rinsed with water (not rubbed with filter paper or boiled linen) and then tested against an air bubble in the bubble machine, with the following results: born-

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ite, 58°; chalcocite, 65°; chalcopyrite, 67°; pyrite, clinging contact; quartz, 0°.

It was observed, during the conduct of these tests, that the polished surface of the bornite, chalcocite and chalcopyrite particles lost its luster upon conditioning. Microscopic examination of these surfaces showed irregular patches of an oily material scattered over the surface. The contact angle formed by the displacement of water by this "oil" at the water-mineral interface was greater than 90°. A microscopic examination of the water suspension of commercial diphenyl sulphide shows dispersed throughout the water a large number of minute globules of the sulphide. When a mineral particle is conditioned in the suspension, these globules settle ($d = 1.1185$) and some of them come to rest on the surface of the conditioning mineral particle. Since phenyl sulphide wets the mineral surface (see below), the settled globules spread over the surface of the mineral. As the globules spread they come in contact with other spreading globules and coalesce, producing thereby the irregular outline observed rather than the circular outline expected of a single globule. To further test this explanation, several attempts were made to filter out the globules and condition the mineral specimens in the filtrate. Filter paper, glass wool, asbestos and cotton were used as filtering media. In all cases the Tyndall cone of the filtrate was weaker and the mineral surface was cleaner. In all cases the contact angle of conditioned chalcocite (the only mineral so tested) was 62° ± .2°. An even cleaner mineral surface is produced by conditioning in the unfiltered suspension if the particle is held with the polished surface downward. In this position the settling globules do not come to rest on the polished surface. Despite this some "oily" patches were observed. These, it is believed, are caused by globules that come in contact with the polished surface during its introduction and withdrawal from the conditioning suspension; moreover, since this suspension is not completely free of currents, globules carried by the currents could have come in contact with the surface during the conditioning period.

Another series of contact-angle tests was made, this time the conditioner was a water suspension of commercial diphenyl sulphide (0.284 grams per liter) and lime (3.4 grams per liter). The concentrations used are again those found in the ball mill. The following results were obtained: bornite, 45°; chalcocite, 64°; chalcopyrite, 0°; pyrite, 0°; quartz, 0°.

Chalcocite conditioned in a filtrate of this suspension gave a contact angle of 61°.

Oiling Tests

A series of oiling tests was run, by placing a drop of commercial diphenyl sulphide on the polished surface of each mineral. The contact angles measured through the water are as follows: bornite, 119°; chalcocite, 112°; chalcopyrite, 98°; pyrite, 40°; quartz, 20°. It was observed that the angles for bornite, chalcocite, and chalcopyrite increased with time; e.g., the angle for bornite after 45 min. was 161°. The angles for pyrite and quartz increase, if at all, very slowly; it is this author's opinion that the angles reported for these two minerals are not true contact angles.

Diphenyl Sulphide a Collector

The inevitable conclusion to be drawn from the facts cited above is that the material purchased under the label of diphenyl sulphide is a collector. According to the chemical theory of flotation, we should expect compound formation between the commercial diphenyl sulphide and the mineral surface. To test this, an attempt was made to produce the reaction product. To a dilute solution of $CuSO_4$ was added some commercial diphenyl sulphide. It was observed that most of the sulphide settled to the bottom of the beaker while some of it formed lenses at the air-$CuSO_4$ solution

interface. The solution-phenyl sulphide interface of the lenses was soon coated with a fine white precipitate. Upon standing overnight it was noticed that the disagreeable odor of the sulphide had been replaced by a pleasant aromatic odor not unlike the odors of diphenyl and phenyl ether. At this time it was recalled, and subsequently confirmed, that a similar pleasant odor was developed in the ball mill after the copper ore was ground in the presence of commercial diphenyl sulphide. By careful manipulation a lens of commercial diphenyl sulphide that had been in 24-hr. contact with a $CuSO_4$ solution may be ruptured and a relatively clean solution-sulphide interface formed. Careful observation shows that this interface does not become coated with the white precipitate. It may be concluded therefrom that the reaction whose product is the precipitate has gone to completion. If the precipitate were due to a reaction between commercial phenyl sulphide and $CuSO_4$, then, since the product is removed by precipitation, the reaction would be expected to proceed until the equilibrium concentration of one or the other of the reactants is attained. This expectation is contrary to fact, hence it is concluded that a reaction does not take place between $CuSO_4$ and commercial diphenyl sulphide but between $CuSO_4$ and some impurity in the sulphide. That the reaction takes place between Cu^{++} and this impurity is indicated by the absence of a precipitate when commercial diphenyl sulphide is added to Na_2SO_4.

Search of Literature

In order to get some indication of the identity of the impurity, a search of descriptions of preparation methods was made in the literature. As a result, it appears that the principal impurities to be expected are thiophenol, diphenyl sulphoxide, diphenyl sulphone and diphenyl disulphide. These compounds have boiling points sufficiently different from that of phenyl sulphide to warrant an attempt at separation by the method of fractional distillation. In order to reduce the possibility of oxidation of the sulphide to the sulphoxide or sulphone, the distillation was carried out in an atmosphere of nitrogen. Since rubber stoppers and tubing contain sulphur compounds and usually are covered with a film of grease, the danger of contamination from these sources was eliminated through the use of an all-glass distillation apparatus. The fractionation was carried out in the usual manner and a sulphide cut boiling at 172°C. at 30 mm. was taken. This product showed that precipitate formation with $CuSO_4$ had a disagreeable odor and when used to condition chalcocite rendered this mineral water-repellent. Failure of the fractionation method to remove the impurity is possibly due to the existence of constant boiling mixtures. At this point the idea arose of using the known reaction of the impurity to remove it. Toward this end, commercial diphenyl sulphide was thoroughly washed with $CuSO_4$ solution in a separatory funnel, dried with anhydrous Na_2SO_4 and distilled as above. A product boiling at 169°C. at 25 mm. was obtained. This product has the aromatic odor previously described, does not react with $CuSO_4$ solution and gives a zero contact angle with chalcocite. When a drop of the material is placed on a polished surface of chalcocite under water, it does not spread. Strict attention must be paid to cleanliness for the success of this experiment, as simple exposure to the atmosphere for a period of several hours will contaminate the product with "grease" from the atmosphere.

Identity of Impurity

An attempt was made to establish the identity of the impurity. Using the methods of Chamot, the crystalline characteristics of products formed by thiophenol, diphenyl sulphoxide, and diphenyl sulphone were

studied at 400 diameters. The test reagents were $CuSO_4$, $AuCl_3$, H_2PtCl_6 and $Pb(NO_3)_2$. A comparison of the crystalline products of the unknown with the known reveals that the impurity in commercial diphenyl sulphide is probably thiophenol. The identification is not complete. In connection with these tests, it was observed that when a drop of commercial diphenyl sulphide was run into a reagent drop of $CuSO_4$ on the slide, a granular precipitate quickly formed on the $CuSO_4$ side of the sulphide-$CuSO_4$ interface. This indicates solution and diffusion of the impurity and the rapidity of formation of the precipitate indicates ionization of the impurity.

Conclusions

1. Diphenyl sulphide per se is not a collector for the copper-bearing sulphide minerals.

2. The claimed collector action of commercial diphenyl sulphide is due to an impurity.

3. The impurity in commercial diphenyl sulphide is possibly thiophenol.

4. A possible function of diphenyl sulphide in flotation with commercial phenyl sulphide is to oil the thiophenol-conditioned mineral particles.

Acknowledgments

The author wishes to acknowledge his indebtedness to A. F. Taggart and N. Arbiter for their stimulating discussion and constructive criticism of this work. In these discussions, ideas are so freely exchanged that paternity is often forgotten. In this respect also, the paper is indebted to them.

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The Mechanism of Collection of Metals and Metallic Sulphides by Amines and Amine Salts

By Nathaniel Arbiter,* Junior Member A.I.M.E., Herbert H. Kellogg† and Arthur F. Taggart,‡ Member A.I.M.E.

(New York Meeting, February 1943)

The experimental work herein described is presented in support of the following broad hypothesis: Conditioning of metals and metallic sulphides by amine collectors involves metathetic reaction at the solid surface between the collector and an oxidized metallic compound, to produce a coating comprising an amine-bearing compound of low solubility in the system. The type of reaction is different in acid and in basic solutions; both types may occur simultaneously in near-neutral pulps.

Experimental

The experiments comprise contact-angle tests with a variety of metals and metallic sulphides and extraction tests on ground chalcocite.

Contact-angle tests were made principally on metals. Covellite paralleled metallic copper in amine solution at pH 8.8 (Table 3) and in alkaline water, but maintained an angle of about 70° in 10^{-3} HCl solution, where metallic copper lost its angle. The tests with anionic collectors (Table 1) indicate that the behavior of metals in general is parallel to that of their minerals in anionic solutions. It is believed that this is generally true with cationic collectors also, but the covellite test is a warning against a broad assumption to this effect. All test specimens were mounted in Transoptic.*

All tests in conditioning solution were preceded by a test in distilled water; cleaning and polishing were continued in every case until the angle in water was zero. Subsequent transfers into and out of solutions and test cells were made in such a way as to prevent exposure of the surface to air.[1]

The amine used was primary laurylamine, Armour and Co., AM-1120. It was used in the form of the hydrochloride, obtained by precipitation with gaseous hydrochloric acid from an ether solution of the amine, and purified by two recrystallizations from ether-alcohol.[2]

Inorganic and other test reagents were standard laboratory grades.

Table 1.—*Contact Angles of Metals Treated with Potassium Ethyl Xanthate or Sodium Oleate*

Metal	Potassium Ethyl Xanthate[a]	Sodium Oleate[b]
Copper	61	93[c]
Platinum	58	94[c]
Silver	61	
Zinc		88
Nickel		85[c]
Cadmium		82
Iron	0	93[c]
Tin	0	89
Magnesium		90
Aluminum	0	95[c]
Lead	43	

[a] Angle in 50 mg. per liter potassium ethyl xanthate.
[b] Angle in 50 mg. per liter sodium oleate.
[c] Angle in water after conditioning in soap solution.

Temperatures were not controlled; the probable range of test-solution temperatures is 65° to 80°F.

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* A transparent plastic sold by Adolph Buehler and Co., Chicago, Ill.
[1] References are at the end of the paper.

Results of the principal experimental work are summarized in Tables 1 and 3 and in Figs. 1 to 3 inclusive. Results of accessory and confirmatory tests are given, together with descriptions of the essential features of the tests, at pertinent places in the discussion.

Nature of Metalliferous Surfaces

The experiments summarized in Table 1 were run for the purpose of making sure that there were metallic ions available in aqueous solution at the surfaces of all the metals tested. It has been shown[3,4,5] that the formation of water-repellent coatings on metalliferous surfaces by means of ethyl xanthates and by oleates involves ionic metathesis resulting in the formation of the xanthate or oleate of the metal involved. Since contact angles characteristic of the anions[6] were obtained for all the metals later tested in amine solutions, it is concluded that there were metal-bearing ions available at the metallic surfaces in all such tests.

It has been proved that the surfaces of sulphide minerals ground in air or in ordinary waters are coated with oxidized-sulphur compounds of the metals.[3,4] The nature of the anions present at the surfaces of the metals in the present test solutions has not been proved, but it is thought that hydroxyl, some carbonate, and possibly some chloride were present when conditioning was done in alkaline solutions, and that hydroxyl, chloride and/or a chlor-metal acidic ion were present in acid solutions.

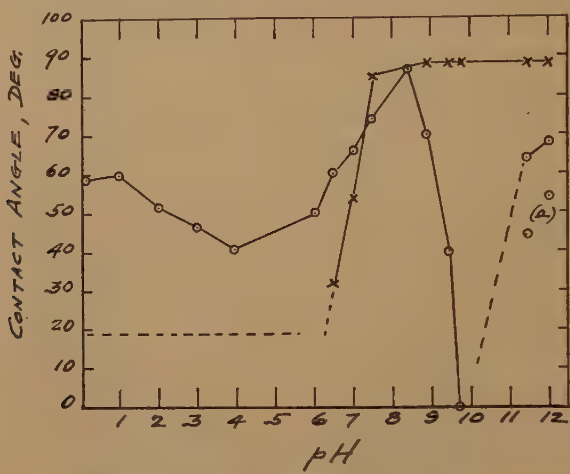

Fig. 1.—Contact angles on platinum.
o indicate points in solution of laurylamine hydrochloride, 25 mg. per liter.
x indicates points in water after 10 min. in amine solution at corresponding pH.
a indicates values in amine solution with lapse of time.

It is postulated, in view of the experiments herein, and of the summary of modern views on the character of metal ions in aqueous solutions presented by Hammett,[7] that the metalliferous ions on the surfaces of the particles tested in the present investigation were hydrated; i.e., were of the general nature of coordination complexes in which the coordinating groups, prior to the collecting reaction, were water molecules.

Amine Compounds in Aqueous Solutions

When an amine salt, such as laurylamine hydrochloride, is dissolved in water, an equilibrium is set up between free amine

and laurylammonium ion according to the following equation:[7]

$$R \cdot NH_3^+ \rightleftharpoons R \cdot NH_2 + H^+$$

It follows that in a solution to which a given amount of amine salt has been added, the relative and absolute quantities of free amine and laurylammonium ion present at any instant depend upon the pH of the solution; that addition of hydrogen ion increases the concentration of laurylammonium ion and correspondingly decreases the amount of free amine, while addition of hydroxyl ion acts conversely.

TABLE 2.—*Estimated Concentrations*[a] *of Laurylamine and Laurylammonium Ions at Different* pH *Values*

pH	Free Amine in Solution, Mg. per Liter	Laurylammonium Ion, Mg. per Liter
1	10^{-6}	25
3	10^{-4}	25
6.1	0.2	25
8	8	16
8.8	$> 8 < 16$[b]	5.5
10	$> 8 < 16$	10^{-1}[c]
11	$> 8 < 16$	10^{-2}[c]

[a] Estimated by assuming that an equilibrium constant K for the dissociation of laurylammonium ion could be calculated from the pH of a solution containing 25 mg. per liter (1.1×10^{-4} mols per liter) by the equation $x^2/(c - x) = K$, where x represents the concentration of hydrogen ion and of dissolved laurylamine and $c - x$ the concentration of laurylammonium ion. From the value of K the concentrations at other pH values below that at which free amine begins to precipitate (between pH 8 and pH 8.8) were approximated by use of the same equation in the form $x(H^+)/(c - x) = K$.

[b] At or about this pH value the formation of a new phase, probably free laurylamine, was indicated by the appearance of a Tyndall cone. The concentration of free laurylamine in solution lies, therefore, between 8 and 16 mg. per liter, which is the value estimated from the equation for pH 8.8 on the assumption of solution.

[c] When the concentration of free laurylamine exceeds its molecular solubility, a separate laurylamine phase forms. The concentration of dissolved molecular laurylamine should be unaffected thereafter by pH increase (Nernst's principle) while the concentrations of laurylammonium ion should vary inversely as the hydroxyl-ion concentration. The values marked [c] are calculated on this basis.

Estimated average concentrations of the two lauryl-bearing groups in solutions of 25 mg. of laurylamine hydrochloride per liter of water at the different pH values tested are given in Table 2.

Collecting Mechanism in Alkaline Solutions

Table 3, second column, shows that of the metals tested Cu, Pt, Ag, Zn, Ni, Cd, Fe and Sn are coated with a hydrocarbon-bearing coating* when conditioned in an alkaline amine solution, while Mg, Al and Pb are not so coated. Column 5 of the same table shows that all of the minerals that become coated form water-stable coordination complexes with ammonia. For most of them corresponding amine complexes have been prepared.[7,9] On the other hand, no water-stable ammonia complexes with magnesium, aluminum or lead are known, in so far as a relatively comprehensive search of the literature has revealed. The fourth column of Table 3 indicates that the coatings on platinum and silver survive transfer to acid solution while those on all the other metals originally coated do not. The behavior on transfer to alkaline solution (column 3) shows that the coatings generally survive transfer from the amine solution and confirm the above indication that acid is responsible for the destruction shown in column 4. Hammett reports that the ammonia-type complex of platinum is stable throughout both acid and alkaline pH ranges and that that of silver is also relatively stable, but that all the others, although more or less stable on the alkaline side, break down in acid.

On the basis of this complete correlation between the formation and stability of the metal-ammonia complexes and the ability of amine to produce hydrocarbon-like coatings on these metals, it is postulated that the coatings are surface compounds in which the cation is the metal-amine complex. Independent tests confirming this hypothesis follow:

* The necessity of a hydrocarbon-bearing coating at a solid surface, if that surface is to be preferentially wet by air in the presence of water and yield a contact angle, has been discussed extensively in the literature,[8] and is accepted herein as established.

Test 1.—Analyses for copper made on solutions with which finely ground chalcocite was leached showed small and equal quantities of copper ion in both water-leach solution and that from an alkaline amine leach. The amine renders chalcocite water-repellent. The contact angle shows that a hydrocarbon-bearing group is abstracted by chalcocite. This test shows that the abstraction is *not* cation exchange. The experimental fact, however, is consistent with formation of a copper-amine complex at the chalcocite surface.

Test 2.—Contact angles induced by conditioning with alkaline laurylamine hydrochloride at metallic surfaces are decreased by ammonium ion in proportion to the concentration of such ion present in the conditioning solution. In a laurylamine hydrochloride solution (25 mg. per liter) the presence of ammonia to the extent of 0.002-N NH_3 (pH 9.4) depressed the angle for copper to 76° and 0.02-N NH_3 depressed the angle to "cling." The water angles did not lift after these depressions, in contradistinction to the behavior of copper in the same range of pH obtained with NaOH as shown on Fig. 2. The implication of this difference is that the ammonia prevented effective formation of metal-amine complexes to the extent that metal-ammonia complexes were formed in accord with mass-action equilibria. The metal-ammonia salts, being soluble, did not cause water repellence at the surface. Fig. 3 taken with Table 2 indicates that the contact angle varies with the proportion of the surface that is amine-coated. Tests reported by Gaudin and Vincent[10] indicate similar behavior with heptoate coatings on calcite. Taggart and Arbiter[11] report the same phenomenon with oleate on a number of minerals. Ammonium ion does not depress the contact angle with amine in acid solutions containing 2 grams per liter of NH_4Cl, where cation exchange is indicated herein to be the mechanism of coating. Hence the depression in alkaline solution is taken to be further evidence of metal-amine complex formation at amine-conditioned metal surfaces.

Test 3.—Baker[12] found that while commercial α-naphthylamine would cause clean granular galena to mirror, the extent

TABLE 3.—*Contact Angles of Amine-conditioned Metals*

Metal	Contact Angles, Degrees[k]			Water-stable Coordination Complexes[d]
	In Amine Solution[a]	In Alkaline Solution[b]	In Acid Solution[c]	
Copper	81	89	[f]	$Cu(NH_3)_x^{+}$ or $^{++}$
Platinum	70	89	88	$Pt(NH_3)_x^{++}$ or $^{++++}$
Silver	45[e]	89	50[e]	$Ag(NH_3)_x^{+}$
Zinc	83[e]	89	0	$Zn(NH_3)_x^{++}$
Nickel	76	89[g]	0	$Ni(NH_3)_x^{++}$
Cadmium	83[e]	89[h]	0	$Cd(NH_3)_x^{++}$
Iron	75[e]	89[i]	0	$Fe(NH_3)_x^{++}$[j]
Tin	65	68	0	$Sn(NH_3)_x^{++}$ or $^{++++}$
Magnesium	[f]	0	0	None
Aluminum	[f]	0	0	None
Lead	[f]	[f]	0	None

[a] Aqueous solution of laurylamine hydrochloride, 25 mg. per liter, raised to pH 8.8 with NaOH.
[b] Particle tested was the same as that tested in the amine solution. It was removed therefrom, rinsed in 10^{-3}N NaOH solution and then tested in 10^{-3}N NaOH solution.
[c] As note [b] except that the rinse and test solutions were 10^{-3} N HCl.
[d] Data from Mellor[6] and from Hammett[7].
[e] Deviations in successive measurements greater than ±2°.
[f] Cling only; no stable contact angle. See del Giudice[1].
[g] Angle falls with time to 77°.
[h] Angle falls with time to 80°.
[i] Rinsed and tested in 10^{-4} N NaOH solution. Angle falls with time to 50° or lower.
[j] No ferric complex reported.
[k] Except where otherwise footnoted the angles reported are averages of several determinations made at different points on the polished surface, differing from one another by not more than ±2°.

of such mirroring decreased as the amine was progressively purified, and that mirroring ceased when the amine, after a number of purifications, was pure white and melted sharply on the reported melting point for the pure substance. He attributed the mirroring to impurities in the commercial amine, probably residual naphthoic acid, but possibly an oxidation product of the amine itself. From the standpoint of the present argument the fact is consistent with the behavior of lead as recorded in Table 3, which, in turn, is a part of the basis already stated for the complex hypothesis.

It is submitted that these tests, totally unrelated except in that all deal with the coaction of amines and metallic ions, constitute ample confirmation of the complex-cation hypothesis, itself based on a cause-and-effect correlation that is most persuasive.

Collecting Mechanism in Acid Solutions

Figs. 1 and 2 show, among other things, that the value of the contact angle produced by amine conditioning in an acid solution is markedly lower than that produced in an alkaline solution. It is proposed that this difference is due, not to differences in the effects of acid and alkaline environments on a coating of the same nature, but to a difference in the nature of the coatings produced in the two environments. It is postulated that the coating in the acidic amine solution is a compound in which the laurylammonium ion is the cation. Direct evidence of cation exchange has not been obtained. Platinum metal in a form with sufficient surface of known character to yield analytical differences of sufficient magnitude was unavailable. Both ground chalcocite and atomized copper dissolve to such an extent in hydrochloric acid solutions of the pH values of the amine test solutions that changes in copper concentration due to the presence of amine in the extracting solution are not great enough to be free of suspicion of analytical error. Indirect evidence, however, is plentiful and is cited below.

1. The contact angle is the same as that obtained with laurylamine hydrochloride on barite and quartz, with both of which

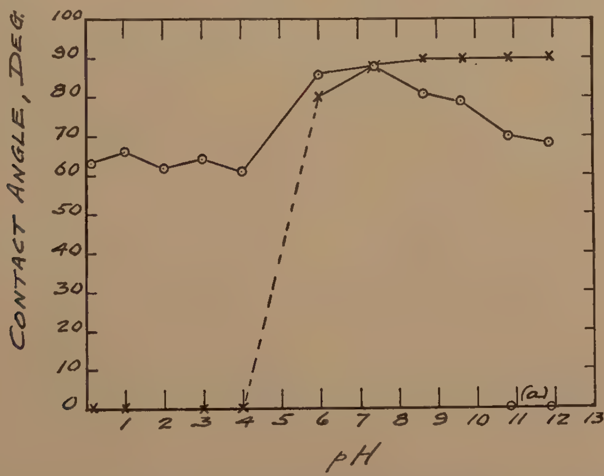

FIG. 2.—CONTACT ANGLES ON COPPER.
o indicate points in solution of laurylamine hydrochloride, 25 mg. per liter.
x indicates points in water after 10 min. in amine solution at corresponding pH
a indicates values in amine solution with lapse of time.

the evidence obtained at Columbia is strongly in favor of cation exchange as the conditioning mechanism.[13] The fact that a contact angle is characteristic of the hydrocarbon-bearing ion was clearly established by Wark and Cox.[14]

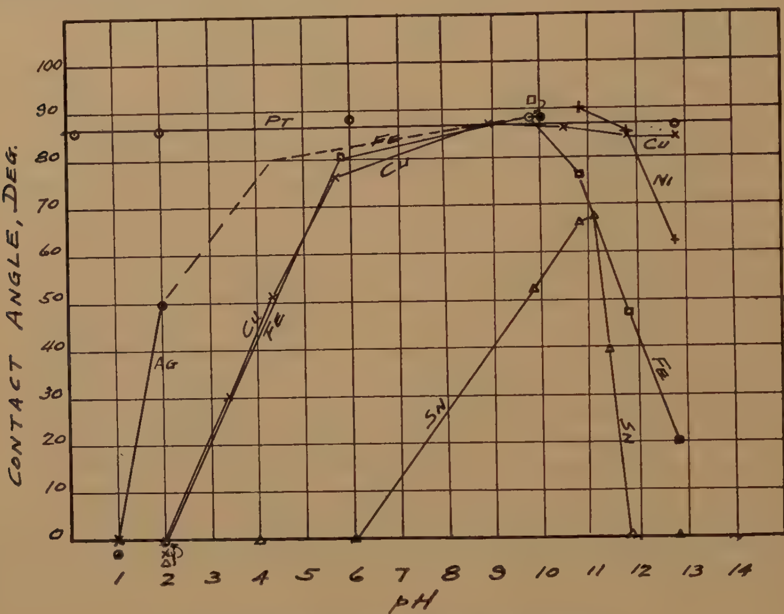

FIG. 3.—CONTACT ANGLES ON VARIOUS METALS.
Amine-conditioned, tested in amine-free solution

2. The curve for platinum in Fig. 3 shows that a platinum surface once conditioned in alkaline amine solution is not affected by an acid environment. Such a surface, according to the hypothesis advanced herein, has a surface coating of a compound in which a metal-amine complex is the cation. On the other hand, Fig. 1 shows that a platinum surface conditioned in acid amine solution loses, or substantially loses, its angle when transferred to water. The first coating is stable; the second is not. Loss or drop in value of angle in water also occurs with barite and with scheelite.

3. It has already been noted that ammonium ion in acid solution does not affect the amine contact angle. It does in alkaline solution. It should in acid also, if the coatings in the two cases were of the same character.

4. When an acid solution of laurylamine hydrochloride is added to a solution of chlorplatinic acid a copious precipitate forms immediately. This precipitate is soluble in water.

All of these facts, independent in their nature, are consistent with the hypothesis that the water-repellent surface coating in an acid amine solution is an amine salt formed by a metathesis involving cation exchange between the amine conditioning salt and an oxidized compound of the metal preexisting at the metallic surface or formed thereupon on immersion in the conditioning solution.

The nature of the metallic surfaces has already been shown to be of ionic character. Anionic metathesis with such surfaces is well established. Cationic metathesis with nonmetallic mineral surfaces is strongly indicated. The contact-angle changes ac-

companying changes in concentrations of amine chloride and of hydrogen ions at the copper surface are all characteristic of an equilibrium reaction at that surface involving a metathesis of the type

$$\mathrm{Cu}A_x + R\mathrm{NH_3}^+ \rightleftharpoons (R\mathrm{NH_3})_y A_z + \mathrm{Cu}$$

where A is an anion, unknown at present but not impossibly a chlorcuprate. Increase in copper-ion concentration should drive such an equilibrium to the left; increase in laurylammonium to the right. Experimental verification of the effect of copper ion was not conclusive because of etching of the surface, which would decrease contact angle independently. Decrease in hydrogen and laurylammonium ions after the formation of $(R\mathrm{NH_3})A$, as by the transfer to water, would tend to cause, first, ionization of the surface compound, then shift of the free amine-laurylammonium-ion equilibrium toward free amine. With the concentration of free amine too low in the resultant water solution to form the copper-amine complex quickly, and probably too low also to form a complete coverage thereof in any case, the contact angle should fall, as it does.

The guess that A in the above equation may be chlorcuprate ion is based in part on the performance of platinum, and in part on consideration of the solubilities of the amine compounds otherwise possible in the copper and platinum systems. Low solubility of laurylammonium chlorplatinate in the presence of excess laurylammonium chloride is indicated above. Laurylammonium chloride is highly soluble in acid solution; neither laurylammonium hydroxide nor carbonate can exist in acid solution. Hence none of the simple anions available at either the platinum or copper surfaces can form a laurylammonium salt insoluble in small concentrations in acid solutions. The drop and disappearance of the contact angle against copper conditioned in alkaline amine solution shows that the copper-amine complex is not stable in acid solutions. The fact that the platinum coating formed in acid amine hydrochloride solution falls greatly in water, compared with the fact that the coating formed in alkaline amine solution persists in water and in strong acid shows that the platinum coating in the two cases is of different character. Shortly, there is in the acid amine solution a hydrocarbon-type coating; it cannot be an amine-complex salt, the weight of the evidence is that it is a laurylammonium compound, and it cannot be such a compound with any one of the simple ions that could possibly be available under the prevailing conditions at either of the metal surfaces. On the other hand, the chlormetallic anions, although not known to form under the conditions prevailing, do form readily under more strongly oxidizing conditions. It is not impossible, therefore, that minute quantities are present in equilibrium generally in aqueous hydrochloric-acid systems containing platinum or copper, and that in the presence of excess laurylammonium ion to form with them the corresponding relatively insoluble chlormetallate, the metal-chloride reaction proceeds and becomes of significant and effective magnitude.

The contact angles intermediate between $\pm 60°$ and $\pm 80°$ as pH changes from ± 5 to ± 8 in Figs. 1 and 2 are consistent with the hypotheses advanced in that they represent the behavior in a transition zone in which both types of surface condition are present. The variation in solutions of low amine concentration in the same pH range (curves X) is what should happen with variation in the percentage of total surface covered by metal-amine complex compound, the water-soluble amine-salt surface compound playing no part in the water repellence.

Effects of pH

The fact that recovery and pH of a flotation pulp are related has been known ever since the early days of froth flotation

Gaudin[15] showed that the optimum pH differs for different collectors and his work suggests a different optimum for different minerals with the same collector. Taggart and Arbiter[11] show that, as far as collection by soaps is concerned, there is a different optimum pH for flotation of each metallic cation, dependent upon the basic character of the cation, and that depression either side of this optimum is proportional to the hydrogen-ion concentration and apparently is due to difference in the degree of ionization of the surface-anchored hydrocarbon compound.

The effects of variation in hydrogen-ion concentration on attachment of air bubbles to amine-conditioned metalliferous particles are indicated in Figs. 1, 2 and 3. The effects are different—at least as far as platinum and copper are concerned—according to whether or not excess amine is present. Thus from Fig. 2, Fig. 3 and Table 3 taken together, it appears that bubble attachment to copper and, presumably, to copper sulphides conditioned and floated in amine solution, will be less effective in acid than in mildly alkaline pulps. This has been, in fact, the experience in mill use of α-naphthylamine for copper ores. High alkalinity, on the other hand, will either depress copper completely or depress it somewhat and require flotation immediately after conditioning. Again, it is indicated that conditioning in a thick alkaline pulp and then diluting to decrease concentration of the amine will increase floatability since, as shown in Fig. 2 and Table 3, such dilution increases contact angle. On the contrary, such dilution in the case of a pulp that is acid during conditioning will result in diminution of floatability. Fig. 1 for platinum is sufficiently like Fig. 2 to indicate similar behavior of platinum minerals.

The reason for the drop in contact angle in amine solutions at high pH is thought to be due to the adsorption (see reference 3) of a second layer, this one of free amine, at a surface already of hydrocarbon nature, as previously developed. This second layer is postulated to be oriented with the NH_2 end of the amine toward the water, and to be more or less complete according to the concentration of free amine, which, in turn, is dependent upon the pH (see Table 2). Such second-layer adsorption is well established for soap films,[16] and produces similar effects on contact angles.[11] The adsorption is attributed to van der Waal's forces between the hydrocarbon groups. The result is to present the water-avid amino groups toward the water and such orientation has been shown[17] to prevent bubble attachment.

Evidence in support of this hypothesis of mechanism in the present case follows. In the first place, when particles with a low angle were taken out of the amine solution at pH values above 8, the liquid peeled off the surface immediately; in fact, as rapidly as the surface of the particle emerged through the surface of the conditioning liquid, it was dry. This dryness is characteristic of particles of high contact angle and indicates complete collector coating by a powerful collector. When this particle was put in water, the contact angle was high (Fig. 1, curve X to the right of pH 8). It would appear that whatever had prevented contact in the amine solution was gone. When the surface was brought into air through an air-water interface, the second amine layer, not being held by valence or ionic forces, but simply by the attraction between like molecular hydrocarbon groups, and attracted to water by the strong hydrating tendency of the amine group, would part from the surface and go into the air-water interface. In the second place, the tendency for the hydrocarbon groups of the amine to aggregate the molecules is seen in its tendency to precipitate as a colloid at low concentrations. Thus a laurylamine hydrochloride solution containing 25 mg. per liter of the hydro-

chloride shows a definite Tyndall cone at pH 8.

An alternative hypothesis is that the depression is due to a flocculation effect involving the coated particles and colloidal free amine. Undissolved free amine in the conditioning solution showed maximum flocculation around pH 10 and rapidly increasing dispersion on both sides of this value. Bankoff[18] has pointed out that gelatin, a hydrophilic colloid, coats a hydrocarbon-coated surface heavily and that such coating is a flocculation phenomenon. A coating of such flocculated amine on a solid surface would not have water-repellent qualities. It would not form on the hydrocarbon-like surface except under flocculating conditions.

Some detailed attention should be given to Fig. 3 for its bearing on the hypothesis of a complex coating ion. The striking features of the curves are: (1) The fact that the composite optimum for all curves lies in the pH range 8 to 10; (2) the broad optima of platinum and silver; (3) the low optimum of tin.

The significance of the persistence of the platinum and silver contact angles in the acid region has already been discussed. The drop shown by the other metals corresponds with the view expressed by Hammett[7] that the hydrogen complex $H(RNH_2)^+$ is more stable in the presence of high hydrogen-ion concentration than are the metal-amine complexes $M(RNH_2)_x^{n+}$. The drop of the nickel curve at the right correlates with the behavior of the nickel-ammonia complex as stated by Hammett;[7] viz., when ammonia is added to a solution of nickel ion, nickel hydroxide $Ni(OH)_2$ first precipitates; further addition of ammonia dissolves the hydroxide and the solution takes on the blue color of the nickel-ammonium ion, $Ni(NH_3)_6^{++}$. Further addition of strong NaOH destroys the blue color and $Ni(OH)_2$ again precipitates. The explanation offered for this final change is that raising the concentration of hydroxyl ion suppresses ionization of the $Ni(OH)_2$ in equilibrium with the nickel-ammonia complex; the latter thereupon dissociates to maintain the concentration of Ni^{++}, which continues to precipitate as $Ni(OH)_2$ as fast as it is available. That a similar phenomenon occurs with iron is indicated by the appearance of a brownish deposit at the conditioned polished-metal surface when it is introduced into solutions of high pH. The hydroxides of silver and of copper are more soluble than those of iron and nickel; that of platinum is unknown, indicating yet greater solubility. Start of the fall of the curve for copper is indicated in the figure. Silver and platinum were not investigated beyond the points recorded. The data are sufficient, however, to support the generalization that maintenance of contact angle in alkaline solution is dependent upon the concentration of hydroxyl ion, the solubility of the metal hydroxide, and the instability constant of the complex.

The decrease of angle with time, cited in Table 3, notes *g*, *h* and *i*, was pronounced in all tests with iron. In one case an 88° original angle fell to "cling" in 15 min. The drop was always coincident with the appearance of the brownish red deposit aforementioned as probably ferric hydroxide. Since only the ferrous ion is credited in the literature with forming metal-ammonia complexes, this is the form of iron that is in equilibrium in solution with the iron of the complex. As this iron oxidizes, the complex must dissociate to maintain the equilibrium. Alternatively the cause of the drop may have been fouling of the surface with ferric hydroxide, due to permeability of the coating of metal-amine compound.

The behavior of tin is not so clearly explicable as that of the other metals, but in so far as published data are available, it is in accord with the complex-ion theory. Thus Mellor[9] states that the tin-ammonia complex is destroyed in "not too alkaline"

solution with formation of stannite and stannate ions, and that it is destroyed in dilute hydrochloric acid with formation of chlorstannate ion.

SUMMARY

Experimental evidence and argument are offered to support the following postulates:

1. When native metals and metallic sulphides are conditioned in alkaline amine solutions, the hydrocarbon-bearing surface layer is a compound in which the cation is a metal-amine complex of the general nature $M(RNH_2)_x^{n+}$ where M is a metal, R a hydrocarbon radical of relatively high-molecular weight and x and n are small whole numbers.

2. When native metals and metallic sulphides are conditioned in acid amine solutions, the hydrocarbon-bearing surface layer is a compound in which the cation is the amine-ammonium ion.

3. The mechanism of both coatings is metathetic exchange. In alkaline solutions free amine displaces water coordinately held by the surface metallic ions in aqueous solution. In acid solutions metathesis is between the metalliferous ion and the amine-ammonium ion.

4. The anion of the surface compound in acid solutions is probably a chlormetallate; in alkaline solutions it may be chloride, hydroxide or carbonate, or a complex of two or three of these groups.

5. Increase in pH beyond a moderate alkalinity in the presence of free amine depresses contact angle by formation of a second-layer amine coating, oriented with amino group outward.

6. Flotation effectiveness of the amine-salt surface compound formed in acid solutions is dependent upon maintenance of an excess of amine-ammonium ion in solution.

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DISCUSSION

(S. J. Swainson presiding)

G. GUTZEIT,* Westport, Conn.—For one who has had the privilege of enjoying the assistantship of the junior author, it is a pleasure to welcome this valuable and interesting paper. It postulates that certain metals and metallic sulphides, when conditioned in alkaline solutions of long-chain aliphatic amines, become coated with an oriented film of a metal amine complex, similar to the corresponding "ammoniates."

Although no quantitative evidence is given in support of this statement, the conclusions being based only on persuasive analogical reasoning and experimental correlations, their validity is extremely probable. An organic amine can be considered as a substituted ammonium molecule, the coordination ability of the nitrogen atom being fully retained. It is well known that these amines form stable complex compounds with the same ions as does ammonia, each amino group being equivalent in coordination capacity to one molecule of ammonia.[19,†]

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* The Westport Mill Research Division, The Dorr Company.

[19] P. Pfeiffer: Organische Molekulverbindungen, Ed. 2. Stuttgart, 1927.

† The presence of functional (polar) groups other than NH_2 and the structure of the

It is well known that the stability of ammoniates is related to the "action radius" of the metal atom. It should also be noted that both amino and ammonia complexes will form only with paramagnetic atoms (i.e., such atoms as are unable to complete their intermediate "valency" electron layer).[20]

Furthermore, the stability of the ammoniates of halogen compounds varies if the metal belongs to a principal or secondary group of the periodic system (contraction effect), and the same rule applies to amino complexes.[21]

The parallelism is thus a very complete one. The same forces and conditions that lead to the formation of ammoniates will produce amino complexes. But, in the chemistry of solutions, a spatial (steric) factor has to be accounted for, which is changed in the case of a mineral surface.[22]

It would, therefore, be interesting to determine the amount of amine abstracted from the solutions, in order to have quantitative data. Anyhow, the mechanism supported by the authors, i.e. metathetic exchange between the (negative) coordinated water molecules and the (negative) amine molecule on the surface cations in alkaline solution, is very likely.

It is furthermore postulated by the authors that when metals and their sulphides are conditioned with laurylamine, in acid solution, the hydrocarbon-bearing surface layer is a compound in which the *cation* is the amine-ammonium ion. It is known that amines, when passing from the alkaline to the acid side (already very near pH 7) are transformed from the weakly dissociated base into the strongly dissociated salt. (Amine films on water, under these conditions, expand considerably with the consequent decrease in lateral adhesion.)

As for the anion (a necessary assumption in order to comply with the "chemical theory" of flotation), the presence of salts of the hydroxo-acids, e.g. chlorcuprate, is suggested. This hypothesis is somewhat doubtful, the formation of such complexes being hardly possible under the prevailing conditions. $Cu[CuCl_2OH]$ and similar compounds are improbable in a weak acid, nonoxidizing solution (except for Sn).

Although the evidence in support of some of the theories presented in this paper is only qualitative, and not always absolutely convincing, the experimental work is excellent and the conclusions are scientifically sound and generally probable, especially for alkaline solutions.

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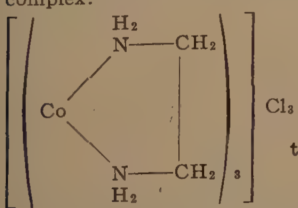
radical has a deep influence on the stability of the complexes. Thus, 2 Aminoethanol forms colored soluble inner-complex compounds with cobalt, gold, manganese, nickel and silver [E. Jaffe: *Ann. Chem. applic.* (1932) **22**]. In the case of diamines, each molecule occupies two coordination positions. For instance, cobalt chloride forms with ethylene diamine the complex:



corresponding to $[Co(NH_3)_6]Cl_3$

. . . As a ring closure follows the coordination of the diamine, the organic cobalt compound is far more stable (toward hydrolysis) than the corresponding ammoniate. This fact suggests the use of long-chain aliphatic diamines as collectors.

[20] G. Gutzeit: *Arch. Sc. phys. et nat. Geneve* (1933) **15**, 409–417.

[21] W. Bilz and H. G. Grimm: *Ztsch. anorg. Chem.* (1925) **145**, 63.

[22] Following A. Magnus [*Ztsch. anorg. Chem.* (1922) **124**, 289 and *Phys. Ztsch.* (1922) **23**, 241], the energy of formation of an ion-molecular complex equals

$$E = pn_1 \cdot e^2/r \cdot d/r \cdot (n - \tfrac{1}{2}n_1S_p \cdot dr)$$

where p is the number of coordinated molecules, n the valency of the central ion, n_1e the charge of the dipole-molecule, ne the charge of the central ion, d the length of the dipole molecule, r the distance from its center to the nucleus of the central ion, and S_p the steric "screening factor" (mutual repulsion) of the p-bound molecules. This latter spatial expression is easy to demonstrate if a simple complex like $[AgI_2]^-$ be considered. Each atom of iodine is attracted by the atom of silver, but repulsed by the second iodine atom. The attraction is e^2/r^2 if e is the elemental electric charge and r the sum of the action spheres (distance) between the nuclei. The second iodine atom, which is supposed to be bound symmetrically, will repulse the first one following $e^2/4r^2 = 0.25e^2/r^2$ (the distance between the two atoms of iodine being $2r$). The factor $S_2 = 0.25$ is called by Magnus "screening factor." If the complex $[AgI_3]^{--}$ is considered, a similar reasoning shows that $S_3 = 0.58$. If there are more than three negative atoms (or polar molecules) coordinated on one central cation, spatial symmetrical dispositions are possible, which change the distance r and therefore S_p. This "screening factor" is always lower for a tridimensional disposition than for a surface. Thus, if four atoms or molecules are coordinated, S_p will be 0.96 for a square and 0.92 for the tetrahedron. With eight coordinated molecules, it will be 1.83 for the hexagon and 1.66 for the octahedron. Consequently, the normal tendency is toward a spatial arrangement, which is impossible in the case of a surface coating.

Oxygen-free Flotation, II—Further Experiments with Galena

By S. F. Ravitz,* Member A.I.M.E.

(New York Meeting, February 1940)

In his excellent book on the Principles of Flotation, Wark[1] makes the following significant statement concerning the theory of flotation:

> Two questions of first-rate importance must be considered . . . First, it must be decided whether the sulfide minerals require a collector in order that they may float, or whether as has been claimed, they possess an inherent floatability and need no collector. Second, it must be decided whether the xanthate type of collector can react with or be adsorbed by the unchanged sulfide minerals themselves, or whether, as many claim, reaction with the collector is dependent upon initial oxidation of the mineral surface. In the theory of flotation there are no two questions of greater importance than these.

Ravitz and Porter[2] presented evidence that galena can be floated without a collector, and even without any reagent, provided oxidation products are thoroughly removed from its surface. They found also that thoroughly cleaned galena floated faster when xanthate was used than when a frother alone or no reagent was used, and that, without a collector, the recovery decreased gradually upon exposure of the mineral to air. Their work, however, has met with considerable criticism.[1(127),3–6] This paper is presented to answer the objections that have been raised, and to present additional data that confirm their results.

Experimental Procedure

The apparatus used to provide oxygen-free conditions for flotation has been described in detail by Ravitz and Porter. An atmosphere of purified nitrogen was maintained within the apparatus; all water and solutions used were freed of dissolved air, and could be transferred where desired by the use of nitrogen pressure; nitrogen was used as the gaseous medium for flotation. The flotation cell was essentially a glass cylinder 15 cm. long and 4.6 cm. in diameter. Near the bottom was sealed a fritted glass filter, which broke up the gas into small bubbles, and 6 cm. above this filter was a side arm through which the froth overflowed. The pulp was agitated by a glass impeller, which entered the cell through a mercury seal. For each test 17 grams of galena and 75 ml. of water were placed in the cell, after which the speed of the impeller and the rate of gas flow were adjusted. Water was added gradually during the test to keep the froth level at the proper height.

The mineral was prepared and cleaned in the following manner. Exceptionally pure galena was crushed by hand on a clean bucking board, screened to −150+200 mesh, and deslimed on a 200-mesh sieve with a fine spray of water. It was then boiled on a hot plate for a number of hours with several changes of the desired cleaning solution, after which both the solution and the mineral were placed in the "mineral flask" of the apparatus. The cleaning solution was drained off, and the mineral was boiled with oxygen-free water

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[1] References are at the end of the paper.

for about five minutes. The water was then drained off, and the mineral was next boiled with oxygen-free cleaning solution. After being boiled alternately with water and cleaning solution in this manner about 20 times, the galena was washed repeatedly with boiling water, the washing being continued a number of times after the drainings gave no test for the cleaning reagents used. Finally, the cleaned galena was transferred as desired from the mineral flask to the flotation cell by means of nitrogen pressure.

The mineral was not permitted to become dry from the time desliming was started until after the flotation tests were made. In cleaning the mineral, the liquid used was drained off until it no longer ran freely from the flask under a slight pressure of nitrogen, and was then immediately replaced with other liquid. The cleaned galena was always kept covered with oxygen-free water, and was transferred to the flotation cell for each test as a mixture of galena and water.

In the original experiments reported by Ravitz and Porter,[2] a saturated solution of ammonium acetate was used for cleaning the galena, which was finally washed with water "until no test for acetate could be obtained with ferric chloride, and only a very faint test with a cupric chloride-sodium chloride solution." The main criticisms of the work have been directed against the use of ammonium acetate. It has been pointed out that lead acetate might still have been present at the surface of the galena, inasmuch as a very faint test for acetate was obtained after washing; that basic lead carbonates, which would not be removed by the cleaning procedure adopted, might have been formed; and that acetamide or other organic compounds with collecting properties might have been present in the ammonium acetate or formed upon boiling the solution.

Shortly after the paper by Ravitz and Porter was submitted for publication, but before it was published, a paper by Gaudin and Wilkinson[7] appeared in which the latter authors pointed out the possibility of the presence of organic impurities in ammonium acetate solutions. For cleaning galena they recommended the use of a solution consisting of three parts of water and one part of a saturated ammonium chloride solution to which sufficient hydrochloric acid had been added to make the pH 1.4. Ravitz and Porter therefore immediately made an oxygen-free experiment as described above, using this cleaning solution instead of ammonium acetate. The galena, which was the same as that used in their previous work (from Joplin, Mo.), was washed several times after the pH of the drained-off water was the same as that of the original oxygen-free water and no test for chloride could be obtained with silver nitrate. Using 0.05 lb. of redistilled terpineol per ton of galena, they found that 97.4 per cent of the galena floated, in excellent agreement with their previous results.*

The galena used in the following experiments had the following analysis: Pb, 86.12 per cent; S, 13.21 per cent; insol., 0.60 per cent; Fe, trace; Cu, none; total found, 99.93 per cent. It had come from Baxter Springs, Kans., and was obtained from Ward's Natural Science Establishment, Rochester, N. Y., in the form of excellent crystals about 2 in. on edge. It was sized, deslimed, and cleaned with a solution of ammonium chloride and hydrochloric acid, as described above.

In each test 0.05 lb. of redistilled terpineol was used per ton of galena, unless specifically stated otherwise. When additional reagents were used, the mineral was first conditioned for 5 min. with the reagent, then the terpineol was added and

* An addendum, describing this experiment, was prepared early in 1934, but never was published.

the test started immediately. Under the conditions of these experiments, 0.01 lb. of reagent per ton of galena was equivalent to approximately 1.1 mg. per liter of solution.

Results

A. Effect of Terpineol Alone.—Complete flotation of galena could be obtained using terpineol alone, provided the mineral was sufficiently cleaned. Whenever flotation was incomplete, the recovery could be increased by further cleaning of the mineral. Flotation was rapid: all the mineral that floated did so during the first minute or two.

Similar results were obtained when air was used instead of nitrogen, or ordinary distilled water instead of oxygen-free water, in the flotation cell. Evidently the galena did not oxidize appreciably during the short time required for each test.

A sample of galena that had been crushed, sized, and deslimed as rapidly as possible, and then tested immediately in the cell without being cleaned, gave a recovery of 55 per cent. Similar samples, tested after being exposed to air for about two weeks, either dry or under water, gave no recovery whatever.

B. Effect of No Reagent.—For a given prepared sample of galena, it was found that substantially the same recovery could be obtained with no reagent whatever as with terpineol alone, although the behavior during flotation was somewhat different. When no reagent was used the bubbles were rather heavily laden at the start of flotation, and were fairly stable (though much less stable than when terpineol was used). As flotation progressed, and the amount of galena in the cell decreased, the bubbles became more and more fragile, and an increasing proportion of them would collapse and drop their load of mineral when they reached the surface. It was necessary to control the speed of the impeller and the rate of gas flow very carefully in order to minimize the collapsing of the bubbles. About 15 min. usually elapsed before flotation ceased, although most of the flotation occurred during the first 5 minutes.

The floatability of the galena with no reagent was strikingly observable during the cleaning operation in the oxygen-free apparatus. The galena was treated in the mineral flask in batches of 100 to 200 grams. When the galena was boiled with water, mineral particles would adhere to the steam bubbles formed and would be carried to the surface. Toward the end of the cleaning operation a large proportion of the galena would rise to the surface of the water in this manner, forming a thick froth. When boiling was discontinued, and the flask allowed to cool, most of the galena would fall back to the bottom of the flask, although the surface of the water would remain covered with a layer of galena particles.

C. Behavior of Products from No-reagent Test.—The following experiment shows definitely that when flotation was not complete there was a significant difference in properties between the particles that floated and those that did not. In a test with no reagent, 17.0 grams of galena floated, leaving a residue of 1.5 grams (91.9 per cent recovery). The residue was immediately washed into a beaker and the floated portion was returned to the cell, where it was found to float completely with no reagent. The residue was then returned to the cell; none of it floated with no reagent, and only a negligible amount (0.15 grams) floated after the addition of terpineol. It floated completely, however, when 0.02 lb. of potassium ethyl xanthate per ton was added.

D. Effect of Xanthate.—Virtually complete recovery of cleaned galena was obtained when 0.05 lb. of potassium ethyl xanthate per ton was used, either alone or with terpineol. Flotation was more rapid with xanthate alone than with terpineol alone; the bubbles were much larger and were more heavily laden with mineral, but

they appeared to be somewhat more fragile. Flotation with xanthate and terpineol together was extremely rapid, being complete in less than one minute.

Uncleaned galena that had been exposed to air for several weeks, and did not float with terpineol alone, floated almost completely with 0.05 lb. per ton each of terpineol and xanthate.

E. Effect of Common Ions.—Neither lead nitrate nor sodium sulphide (measured as Na_2S) had any effect on the flotation of cleaned galena at concentrations of 0.1 lb. per ton or less. Sodium sulphide had a noticeable depressing effect at very high concentrations, the recovery falling to 68 per cent at 10 lb. per ton. This may have been due, however, to the high concentration of hydroxyl ion.

F. Effect of Oxidizing Agents.—Cleaned galena was completely depressed by potassium dichromate at a concentration of 0.1 lb. per ton, and by hydrogen peroxide at a concentration of 0.5 lb. per ton.

Some cleaned galena was conditioned for 1 min. in the flotation cell with 1.0 lb. of hydrogen peroxide per ton, the solution was then drained off, and the mineral was washed several times with oxygen-free water. A test was made with 0.05 lb. of terpineol per ton, but none of the galena floated. The galena was then leached with hot ammonium acetate solution. The leach solution gave a definite test for sulphate, indicating that lead sulphate had been formed as a product of the oxidation of the galena by hydrogen peroxide.

G. Effect of Sodium Arsenate and of Sodium Phosphate.—Neither sodium arsenate (Na_3AsO_4) nor sodium phosphate (Na_2HPO_4) had any effect upon the cleaned galena at a concentration of 1.0 lb. per ton. However, uncleaned galena that floated completely with 0.05 lb. per ton each of terpineol and xanthate after being exposed to air for several weeks (see section *D*) was completely depressed when tested under the same conditions after being conditioned for 5 min. with either sodium arsenate or sodium phosphate.

H. Effect of Starch.—As little as 0.01 lb. per ton of arrow-root starch completely depressed cleaned galena.

I. Effect of Miscellaneous Organic Compounds.—Tests were made after adding one drop (about 10 lb. per ton) of acetone, ethyl alcohol, petroleum ether, or ethyl ether to the cell. There was little or no effect with any of these reagents.

J. Contact-angle Experiments.—A 1-in. cube of galena was carefully cleaned with a solution of ammonium chloride and hydrochloric acid, as described above for the sized galena. No contact could be observed when a captive air bubble was pressed against the surface of the specimen submerged under oxygen-free water, ordinary distilled water, or a solution of terpineol. Definite contact was observed, however, in xanthate solution; the angle was not measured, but appeared to be in the neighborhood of 60°.

Discussion of Results

As mentioned, the main objections to the work of Ravitz and Porter[2] were raised against the use of ammonium acetate for cleaning galena: (1) because the cleaning with ammonium acetate might still leave oxidized lead compounds at the surface of the galena, and (2) because ammonium acetate might introduce impurities that would act as collectors. The substitution of the solution of ammonium chloride and hydrochloric acid overcomes these objections. Nevertheless, it is difficult to see how they can apply even to the results obtained with ammonium acetate. The objections fail to account for the following experimental facts reported by Ravitz and Porter and confirmed in the present investigation:

1. Aged galena could not be floated without xanthate.

2. As cleaning progressed, more and more of the galena could be floated with

terpineol alone or with no reagent, until eventually all the galena could be floated.

3. The floatability of the cleaned galena gradually decreased upon exposure to air.

These facts show clearly that the floatability of galena improves as oxidation products are removed, and becomes poorer as oxidation products are formed. They indicate, therefore, that oxidation products, instead of being necessary for the flotation of galena, actually hinder the flotation. This is confirmed by the depressing effect of potassium dichromate and hydrogen peroxide, which form coatings of oxidized lead compounds at the surface of the galena.

The results of the tests with sodium phosphate and sodium arsenate (section *G*) provide further evidence that the cleaned mineral was free from oxidation products. The following seems to be the logical interpretation of these results. The aged galena is coated with natural oxidation products, which make it incapable of floating without a collector. The addition of xanthate enables the mineral to float, presumably by the formation of lead ethyl xanthate. When the mineral is first treated with sodium arsenate or sodium phosphate, however, the very insoluble lead arsenate or lead phosphate is formed, with which the xanthate cannot react (at least not sufficiently at the low concentration of xanthate used, 0.05 lb. per ton or 5.5 mg. per liter); the mineral, therefore, cannot float, even when xanthate is added. Sodium arsenate and sodium phosphate, on the other hand, have no effect upon cleaned galena because no oxidation products are present for them to react with, and they cannot react directly with lead sulphide.

Contaminants

Although it has been shown that oxidation products are not essential for the flotation of galena, there remains the possibility that flotation without the addition of a collector may have been due to organic contamination. The possible sources of contaminants, and the steps taken to guard against them, will therefore be discussed briefly.

Nitrogen.—Because of the exceedingly low temperature at which nitrogen is distilled, it is hardly likely that organic impurities could arise from the air from which the nitrogen was originally obtained. It is conceivable, however, that minute amounts of organic vapors may have gotten into the nitrogen from the machinery used in preparing it from liquid air, compressing it, etc.

Glassware.—It is highly improbable that the glassware, which was cleaned with a hot solution of sulphuric acid and potassium dichromate before being used, was a source of contamination.

Rubber.—Although all rubber parts used were first boiled for an hour in strong sodium hydroxide solution, it is conceivable that organic substances might have been dissolved from the rubber during the experiments. To minimize contamination from this source, liquids that had been standing in contact with rubber were flushed out of the system at the beginning of each series of tests. After it had been found that distilled water could be used in the flotation cell instead of oxygen-free water (see section *A*), all reagent solutions were made up outside the apparatus with freshly boiled distilled water and were poured directly into the cell from a graduated cylinder or pipette. The only liquids, therefore, that flowed through rubber connections were oxygen-free water and oxygen-free ammonium chloride-hydrochloric acid solution.

Cleaning Solution.—Chemically pure ammonium chloride and hydrochloric acid were used for cleaning the mineral.

Galena.—Perhaps the most serious source of contamination was the galena itself, since it was not known what handling it had undergone from the mine to the laboratory. For this reason large specimens were used, so that the surface area was increased

several thousand fold upon crushing to $-150+200$ mesh. In addition, several tests were made with specimens that had been washed thoroughly with acetone before being crushed; the results were not affected.

The experimental facts show that contaminants, if in any way responsible for the results obtained, would have to be of such a nature that they could exert a collecting effect only upon unoxidized galena surfaces and not upon surfaces coated with oxidation products. They would therefore have to be quite different from any substance known to be a collector for ordinary galena. Moreover, no contact could be obtained between a captive air bubble and a cleaned specimen of galena in water. Wark[1(74)] and Taggart[3(370)] regard any tendency of a bubble to adhere to a galena surface as an indication of contamination. Lack of adhesion therefore may be considered indicative of lack of contamination, although not necessarily as proof.

Wark[1(127)] states that in Ravitz and Porter's experiments "the galena was repeatedly allowed to dry during the cleaning process, and . . . this would influence the results." This statement is due to a misunderstanding, for the galena was not allowed to become dry after the cleaning was started. Nevertheless, even if it had been, the tests would still have shown that oxidation products are not necessary for the flotation of galena.

Contact Angle

Wilkinson[4] was unable to obtain contact between a bubble of hydrogen and a piece of galena cleaned with the ammonium chloride-hydrochloric acid solution in an atmosphere of hydrogen, and concluded from this that Ravitz and Porter's results were due to incomplete removal of ammonium acetate and oxidation products dissolved in it. Similarly, Wark and Sutherland[1(127)] were unable to obtain contact between an air bubble and galena cleaned with sodium chloride, ammonium chloride, or ammonium acetate. These results are in accord with the contact-angle experiments of the present investigation, in which no contact could be detected between an air bubble and a piece of galena cleaned with the solution of ammonium chloride and hydrochloric acid. Yet $-150+200$-mesh galena, cleaned in the same way, could be floated completely without any reagent. Apparently, then, galena can be floated under conditions in which it does not exhibit a detectable contact angle. This conclusion has been reached also by Gaudin,[8] who presents considerable evidence to support it. One or more of the following reasons may account for the fact that cleaned galena can be floated without a collector, although it does not form a measurable contact angle with air:

1. As Gaudin points out, the contact angle may be adequate for attachment, and yet be too small (less than about 10°) to be detected with certainty by present methods.

2. Gaudin also points out that flotation seems highly dependent upon the angularity of the particles. Contact angles are usually measured by pressing a captive bubble of air against the plane surface of a large mineral specimen. Flotation, on the other hand, is concerned with small bubbles and still smaller mineral particles, so that the corners and edges of the particles may play an important part. In this connection, some results obtained by Wark[1(56)] with solutions of cetyl trimethyl ammonium bromide are significant. He found that in a solution containing 50 mg. of the reagent per liter, contact could not be obtained directly between an aged air bubble (½ min. old) and the surface of a polished specimen of galena, but could be effected at the sharp edges of the specimen and be retained as the bubble was slid across the flat surface.

3. The force with which bubbles and particles collide in a flotation cell may

cause the gas to replace water at the surfaces of the particles, whereas this would not occur when a large bubble is merely pressed against the surface of a large mineral specimen. The possibility of force of collision being an important factor is supported by some experiments made by Wark and Sutherland.[1(195)] Using a solution in which contact could not be effected (Aerofloat 25, 25 mg. per liter; pH above 8.5) they obtained flotation of galena in a cell with mechanical agitation (shaken stoppered cylinder), but not in a miniature pneumatic cell.

Summary

1. Further work on the oxygen-free flotation of galena has confirmed the conclusion that products of oxidation are not essential for the flotation of galena. Under the conditions of the experiments, galena thoroughly cleaned with a solution of ammonium chloride and hydrochloric acid could be floated with no reagent, with terpineol alone, or with ethyl xanthate alone, the rate of floating increasing in the order given. Uncleaned galena, exposed to air for a few weeks after crushing, could not be floated without a collector.

2. Hydrogen peroxide, potassium dichromate, and particularly starch, were found to be strong depressors for cleaned galena in the absence of a collector.

3. At moderate concentrations sodium sulphide, lead nitrate, sodium arsenate, disodium phosphate, acetone, ethyl alcohol, petroleum ether, and ethyl ether had no effect upon the flotation of cleaned galena in the absence of a collector. Sodium arsenate and disodium phosphate, however, completely depressed aged galena in the presence of 0.05 lb. of potassium ethyl xanthate per ton.

4. No contact could be detected between an air bubble and cleaned galena in water.

5. The matter of organic contamination is discussed. Although the possibility of contamination is not entirely eliminated, it is concluded that it is improbable that the results can be attributed to contaminants.

Acknowledgment

The author gratefully acknowledges the contributions by R. R. Porter, Wallace V. Peck and Ralph Huber, former Research Fellows in the Utah Engineering Experiment Station, University of Utah, to the experimental work described in this paper. He also wishes to express his thanks to A. M. Gaudin, Richards Professor of Mineral Dressing, Massachusetts Institute of Technology, for reviewing the manuscript and offering his helpful criticism.

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DISCUSSION

(*H. R. Hanley presiding*)

J. M. Patek,* Rochester, N. Y.—Mr. Ravitz says that he found that clean galena that floated readily failed to show any contact angle in the captive-bubble test.

In a publication by the present writer,[9] it was concluded that the captive-bubble test

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* Mining Engineer and Geologist.

[9] J. M. Patek: Relative Floatability of Silicate Minerals. *Trans.* A.I.M.E. (1934) 112, 486.

could not be a measure of the wettability of a clean mineral surface, but only a measure of the degree of contamination by an organic substance. The writer's contact-angle measurements seemed to show a contact angle of between 15° and 70° with water on a dry mineral surface. A contact angle of over 90° would be necessary to cause the water to recede when contacted by a captive bubble of air. Only an organic substance would show a contact angle of over 90°. The captive-bubble test apparently measures the degree of completeness of an organic coating on a mineral surface, not the wettability of the mineral surface itself. A clean mineral surface, which is expected to have a contact angle of less than 90°, should show no contact angle in the captive-bubble test.

The conclusions above were arrived at largely through theoretical deductions. The writer is now pleased to see that Mr. Ravitz has verified these deductions by experimentation some six years later.

Differential Flotation of an Arsenical Quicksilver Ore

By Maurice Rey, Member A.I.M.E., and H. Brevers

(New York Meeting, February 1941)

Through circumstances connected with the war, the senior author lost his records, therefore it has been impossible to include numerical data in this paper.

The arsenical quicksilver ore investigated had been roasted, with poor metallurgical results. The vapor-tension temperature curves of mercury and arsenious trioxide are similar, and the arsenical compound hampers the coalescence of mercury drops in the condenser and is the cause of large stack losses. The ore was a gray limestone carrying small amounts of carbon and mineralized with cinnabar and two arsenic minerals, realgar (As_2S_2) and arsenopyrite (FeAsS). The proportion of realgar often exceeded that of cinnabar; the proportion of arsenopyrite was smaller. The grade of the ore varied between 0.8 and 4 per cent Hg. The problem was to make a high-grade mercury concentrate, as free from arsenic as possible.

The water used throughout the testing was hard, having a pH of about 8.

Flotation of Realgar

Realgar showed a very strong tendency to float. It formed a film at the surface of the pulp and floated in abundance with only pine oil. Therefore an attempt was made to float the realgar before the cinnabar, but this was not very successful; the percentage of mercury in the realgar concentrate could never be brought below 2 or 3 per cent. Cinnabar slimed easily and floated with pine oil, so that some of it always accompanied the realgar. There was also the question of the cinnabar-arsenopyrite separation.

On further investigation it was found that realgar can be floated not only with pine oil but even better with a combination of hydrocarbons and pine oil. Washing the ore with ether revealed that it contained hydrocarbons, which helped to float the realgar. In this connection it belongs to the same class as carbon, sulphur and talc. It can be floated also with a xanthate, when activated by a heavy-metal salt.

The conditions favoring the two types of flotation are opposed, as activation by a copper salt facilitates xanthate flotation (in fact, is even necessary to it), whereas it hinders pine oil and petroleum flotation. One type of flotation can be termed nonpolar and the other polar, a distinction that has been stressed in a previous paper.[1]

Depression of Realgar and Flotation of Cinnabar

Many chemicals were tested in an attempt to float the cinnabar and depress the realgar. Finally starch rendered soluble with caustic soda was tried, also several grades of commercial soluble starch, and dextrin was found to be the best and the most convenient reagent. Several of these starch preparations have a strong flocculating effect on the pulp, which is a distinct disadvantage. Dextrin, with its low flocculating power, is excellent.

It is noteworthy that starch preparations are also depressants for carbon and talc, so that the fact apparently has a general

Manuscript received at the office of the Institute Sept. 3, 1940. Issued in Mining Technology, January 1941.

[1] M. Rey: Contribution à la théorie de la Flotation. Congres Int. des Mines, de la Met. et de la Geol. App., Paris, 1935.

significance. The adsorbing properties that make for the adsorption of frothers and hydrocarbons are also responsible for the adsorption of the starch colloid.

However, even this was not sufficient to depress the realgar—it was also necessary to activate the cinnabar. In many ores, cinnabar floats with xanthate collectors without any activation. In the present tests it did not do so. The reason for this was not apparent for a long time and depression by the starch or dextrin used was considered to be the cause, until it was discovered that hydrogen sulphide, generated during the grinding of the ore, was responsible.

Hydrogen sulphide and sodium sulphide are known to be depressors of sulphides, and this action has been studied by Wark and Cox in their series of papers on the principles of flotation.[2]

We found that this depressing action was particularly effective on cinnabar, which is natural in view of the extremely low solubility of mercuric sulphide. The solubility product of this compound is known to be one of the smallest among metallic sulphides and it is much smaller than that of lead or copper sulphide.

Hydrogen sulphide was generated during the wet grinding of the ore by the alteration of realgar according to a reaction that probably is

$$As_2S_2 + 2H_2O + \tfrac{1}{2}O_2 = As_2O_3 + 2H_2S$$

The fact can be ascertained easily by a silver or lead acetate test.

It was noticed that wet grinding had some effect on the flotation of realgar and was favorable to its depression by soluble starch or dextrin.

As a matter of course, copper sulphate was at first used as an activator of cinnabar, and fair results were obtained using potassium xanthate as collector and pine oil as frother.

The mercury-arsenic separation, however, was not entirely satisfactory because arsenopyrite tended to rise toward the end of the float and contaminated the concentrate. Cyanide could not be used to keep down arsenopyrite because it destroyed in part the activating effect of copper sulphate on cinnabar.

A search was then made for an activating reagent that would be better than copper sulphate, and a much more powerful and selective one was found in mercuric chloride. The reagent consumption could be reduced to 50 grams per ton instead of 200 or 300; the froth was more heavily mineralized and, most important, the selection between cinnabar and arsenopyrite was improved.

A further improvement was brougth about by the use of a small amount of cyanide, which this time did not cause any trouble, apparently because of the extremely strong activating effect of the mercuric ion on cinnabar.

The final combination of reagents arrived at was about as follows:

Reagent	Grams per Ton*
Dextrin	250 to 400
Mercuric chloride	50
Potassium cyanide	25
Potassium xanthate	50
Pine oil	50

* Probably metric ton.—Ed.

The mercuric chloride was added either before or after the xanthate. Arsenopyrite did not float unless an excess of activator and of collector was used. From a 1 to 3 per cent Hg ore, containing 2 to 4 per cent As, concentrates assaying 40 to 50 Hg and 2 to 3 per cent As were obtained after one cleaning operation.

The question may well be raised why the activator did not bring forth the flotation of realgar with the cinnabar. Aside from the depressing action of dextrin, the case may be considered as one of preferential activation helped apparently by the fact that hydrogen sulphide is generated precisely at the surface of realgar.

[2] I. W. Wark and A. B. Cox: Principles of Flotation, IV. *Trans.* A.I.M.E. (1939) 134, 7.

A peculiar observation, which appears worth mentioning, is that under certain conditions of excess activator and collector, the froth became so brittle as to be unmanageable. Tests run with and without mineral in the pulp showed that this was caused by the presence as free particles of the precipitate formed by the interaction of mercuric chloride and potassium xanthate. The addition of cyanide dissolved this precipitate and increased the stability of the froth, so that aside from its depressing effect on arsenopyrite, cyanide had a beneficial effect on the frothing characteristics of the pulp.

The fact that cyanide was capable of dissolving the free xanthate compound without impairing the xanthate film at the surface of the mineral is a further proof, if one were needed, that conditions inside a flotation pulp and at the surface of the contained minerals are different. The matter is of interest from the point of view of flotation theory.

Summary and Conclusions

Realgar behaves like a nonpolar mineral similar to carbon, sulphur and talc. Its strong tendency toward flotation can be effectively counteracted, even in the presence of small amounts of hydrocarbons, by the addition of soluble starch or dextrin. The latter is particularly effective because of its low flocculating power.

Cinnabar, which usually is collected by xanthates, does not respond to xanthate when depressed by hydrogen sulphide. It can then be activated by heavy metal ions and among these mercuric ions are the most effective; they activate the mineral even in the presence of small amounts of cyanide. Cyanide can thus be used to depress arsenopyrite and make a good cinnabar-arsenopyrite separation. A cinnabar-pyrite separation could probably be effected in the same way. The addition of cyanide at the same time counteracts a peculiar tendency of the quicksilver-xanthate compound to embrittle the froth.

The case considered here is an example of the efficiency of flotation technique. From a natural tendency toward the flotation of realgar and the depression of cinnabar, due to the presence of hydrocarbons and hydrogen sulphide in the ore, conditions were changed over, by proper reagent additions, to the flotation of cinnabar and depression of realgar. At the same time, a good cinnabar-arsenopyrite separation was effected.

DISCUSSION

(C. H. Benedict presiding)

C. A. Heberlein,* New York, N. Y.—The results stated by Rey and Brevers are so startling that they are of great interest, as the concentration of a low percentage of cinnabar by differential flotation yields a high-grade concentrate in the presence of either arsenical or iron pyrite.

The authors say: "Cyanide can thus be used to depress arsenopyrite and make a good cinnabar-arsenopyrite separation. A cinnabar-pyrite separation could probably be effected in the same way."

Corrosive sublimate at the present price of mercury would be a very expensive reagent but the expense would be justifiable if a high rate of concentration could be obtained with a high extraction. On my request, special tests were made in the Punitaqui mill in Chile along the line suggested by the authors, without any improvement of either grade or extraction.

As the recovery of the cinnabar in a complex sulphide of chalcopyrite and pyrite gave unsatisfactory results, I had tests made at the testing laboratory of the American Cyanamid Co. at Stamford, Conn. As these tests turned out to be highly satisfactory, not only at Stamford but later at the mine, they are presented here.

American Cyanamid Test

Feed	Per Cent
SiO_2	66.54
Al_2O_3	2.08
CaO	0.70
MgO	0.00
Alkalis	1.78
Fe	15.63
Cu	0.44
S	9.42
Hg	0.45
	Oz. per Ton
Ag	0.95
Au	0.151

* Consulting Engineer.

Screen Analysis of Tailings, Per Cent

Plus 65	0.0
Plus 100	1.32
Plus 150	20.75
Plus 200	19.85
Minus 200	58.08

The pulp had a density of 22 and a pH of 7.8. Total time of conditioning was one minute. The reagent added was Texas fuel oil, 0.116 lb. of 33.7 Bé. gravity containing 0.9 per cent sulphur.

Frother, 0.062 lb. of No. 5 pine oil.

Rate of concentration of mercury was 25.7:1.

Metallurgical Results

Product	Assay					Distribution, Per Cent			
	Weight, Lb.	Au, Oz.	Ag, Oz.	Hg, Per Cent	Cu, Per Cent	Au	Ag	Hg	Cu
Feed	100	0.151	0.95	0.45	0.44	100	100	100	100
Hg concentrates	3.89	2.08	20.24	10.80	9.58	53.65	83.31	93.33	84.15
Fe concentrates	15.46	0.40	1.02	0.09	0.35	40.98	16.69	3.09	12.21
Tailings	80.65	0.01	0.00	0.02	0.02	5.37	0.00	3.58	3.64

TESTS AT THE MINE IN CHILE

In the tests recorded below, no attempt was made to extend the recovery beyond the selective flotation of the mercury.

Test No. 1, Using Anglo-Persian Fuel Oil and No. 5 Pine Oil

	Concentrate	Extraction, Per Cent
Au	72 grams per ton	57.3
Cu	10.6 per cent	89.2
Hg	9.7 per cent	94.3

Test No. 2, Diesel Oil and No. 5 Pine Oil

	Concentrate	Extraction, Per Cent
Au	120. grams per ton	54.2
Cu	17.1 per cent	80
Hg	16.2 per cent	87.7

These tests were followed up in the mill at full capacity of 400 tons per day, using fuel oil and No. 5 pine oil:

	Feed	Concentrate	Extraction
Hg, per cent	0.27	7.45	84.0

The results thus far obtained, using fuel oil and No. 5 pine oil in the laboratory and in the mill are very encouraging to selectively float a high-grade mercury concentrate of 10 per cent and better, with a high extraction. It is intended to condition the tailings from the mercury flotation with lime to recover the rest of the gold and copper. A short conditioning and a neutral pH seem to be necessary for a selective flotation of the cinnabar.

Flotation of Barite from Magnet Cove, Arkansas

By James Norman,* Junior Member A.I.M.E., and Benjamin S. Lindsey†

(New York Meeting, February 1941)

Barite ($BaSO_4$) is the most important industrial barium mineral from the standpoint of quantity consumed. In 1938 the amount was 365,000 tons. Its uses are numerous, some of the more important being in the production of pigments, as an inert filler in a wide variety of products, as a weighting material in oil-well drilling muds, in the manufacture of some kinds of glass, and in the production of barium chemicals.

The barite studied in this investigation occurs as the massive variety in a shale replacement formation near Magnet Cove, Ark. It is intimately mixed with quartz and small amounts of iron oxide and residual shale. Complete liberation of barite and quartz is not obtained when the rock is ground to 100 per cent through 325 mesh. The crude rock with which all test work was done assayed 85.1 per cent $BaSO_4$, 11.11 per cent SiO_2, and 2.85 per cent R_2O_3. Its specific gravity was 4.03. It was found in the course of the investigation that the barite contained a small amount of uniformly distributed carbonaceous material, so intimately dispersed that it could not be seen with the microscope, but gave a definite gray cast to the barite concentrates. The color of the barite in polished sections of the crude rock ranged from gray to almost black, while finely ground powder was a reddish gray. The reddish hue was due to the iron oxide present.

In 1939, under the supervision of Oliver C. Ralston,* the Bureau of Mines, in cooperation with the Milwhite Company, of Houston, Texas, undertook to study the flotation of barite from Magnet Cove, Ark. Flotation of several southern barite ores had already been investigated by the Bureau of Mines.[1] The Magnet Cove barite was unique, however, in that it had to be ground to such an extremely small particle size for liberation. The flotation of nonmetallic minerals in the presence of a large amount of slime is usually very difficult, and when done is an innovation calling for conditions different from those under which granular material is separated.

Laboratory Flotation Work

Preliminary investigation showed that the flotation reagents recommended by O'Meara and Coe[1] for separating barite and siliceous material were suitable, but much larger quantities of reagents were required to separate such finely divided material. Moreover, it was found that careful control of the frothing was necessary. When oleic acid and sodium silicate were used a voluminous froth was observed at the beginning of a test, and toward the end there was not enough froth to effect complete recovery of barite. The addition of pine oil as a frother gave proper frothing for the duration of the test, and the substitution of coconut fatty acid (crude lauric acid) for oleic acid produced a more heavily burdened froth. Two different paper-mill fatty-acid products were tested

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[1] References are at the end of the paper.

and found to be inferior to both oleic and lauric acids, as their selectivities were lower than that of either of the acids.

the minus 28-mesh crude, allowed the production of concentrates of around 4.30 sp. gr., with a recovery of about 80 per

TABLE 1.—*Details of Two Batch Tests*

BATCH TEST (250 GRAMS) ON MINUS 28-MESH PEBBLE-MILLED BARITE

Reagents	Quantity, Lb. per Ton			Results of Separation					
	Rougher[a]	Cleaner 1	Cleaner 2	Products	Weight Per Cent	Analysis, Per Cent $BaSO_4$[b]	SiO_2	R_2O_3	Specific Gravity
"S" brand sodium silicate	4.6	1.8	1.8	Cleaner concentrate	75.9	98.43	1.42	0.44	4.36
Coconut fatty acid	1.8			Middlings (combined)	12.8	68.79	22.03	6.10	
Pine oil	0.25			Tailing	11.38	46.89	41.58	9.56	
				Feed	100.0	85.1	11.11	2.85	4.03

BATCH TEST (1000 GRAMS) ON RAYMOND-MILLED PEBBLE-GROUND BARITE

Reagents	Quantity, Lb. per Ton			Results of Separation		
	Rougher[a]	Cleaner 1	Cleaner 2	Products	Weight Per Cent	Specific Gravity[c]
"S" brand sodium silicate	10.8	0.54	0.54	Cleaner concentrate	74.0	4.37
Coconut fatty acid	3.22			Middlings (combined)	9.8	
Pine oil	0.141	0.094	0.094	Tailing	16.2	
				Feed	100.0	4.05

[a] Rougher reagents conditioned for 10 min. before floating.
[b] Recovery of $BaSO_4$ in cleaner concentrate, 87.8 per cent.
[c] The specific gravity of the concentrates was used for most evaluations.

Preliminary tests were run on crude material prepared in two different ways. First, the crude lump was crushed dry to pass a 28-mesh screen, then wet-ground in a 1-gal. pebble mill. Second, a crude Raymond-mill product, 98 per cent through 200 mesh, was wet-ground in the pebble mill. The optimum grind in the pebble mill was 1 hr. for the minus 28-mesh material and 15 min. for the Raymond-mill product. One to two pounds of "S" brand sodium silicate per ton of charge was used in the grind to secure deflocculation. In no instance was the flotation feed deslimed. All the product from each grind would pass a 325-mesh screen. Details of two batch tests are given in Table 1.

It was shown in other tests that when a coarser feed from less severe grinding was used inferior separations were obtained. Pebble-mill grinds of 10 to 30 min., with

cent of the feed in the concentrates. Batch tests, and later continuous test-plant oper-

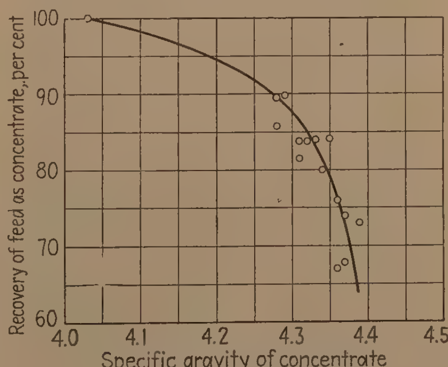

FIG. 1.—LABORATORY BATCH FLOTATION TESTS (250 GRAMS).

ation, showed that while good separation of the dry-ground Raymond-milled product could be obtained, the separation was considerably improved by a short wet grind.

The proportion of the feed recovered in the concentrates depended on the amount of grinding and the quantities of reagents used but bore a definite relationship to the specific gravity of concentrates. This is shown from results of several tests by the curve in Fig. 1. This relationship is thought to be due to middling particles, seen microscopically, even at the finest grind.

Continuous Testing

After completion of laboratory batch tests, approximately 1 ton of Raymond-milled crude rock was treated in several continuous tests with a continuous 6-cell (12-in.) flotation unit. The flowsheet was the same in all runs, with the one exception noted below.

The dry-ground crude material was fed to the ball mill by means of a belt feeder at the rate of 60 lb. per hour. All the sodium silicate used was added here and ground with the barite. Water was added at the rate of 60 lb. per hour, and the ball-mill discharge was 50 per cent solids. There was some fluctuation in the consistency of the discharge, owing to fluctuation in the pressure of the water supply. Iron balls were used in the mill, and no attempt was made to prevent contact of the pulp with iron throughout the whole circuit. None of the slime was removed from the pulp. The ball-mill discharge was pumped to a conditioner, where coconut fatty acid was added. The conditioner was of the type ordinarily used, consisting of a sheet-iron drum with a splash overflow. Agitation was produced by a ½-hp. portable electric mixer. From the conditioner, the pulp overflowed to the 6-cell flotation unit. It was introduced at the third cell of the unit, and flow of concentrate and tailing was countercurrent cell to cell. With this arrangement there were four rougher and two cleaner cells. Adequate dilution of the feed was obtained from froth-breaking sprays in the concentrate launders. Concentrate and tailing were collected in separate thickening cones, and the thickened products were removed alternately with a diaphragm pump. Pine oil was added in No. 4 cell.

Duplicate 15-hr. runs were made with the flowsheet described above, to see whether there would be a building up of middling particles in the cells and a subsequent lowering of grade of concentrate. This was shown not to occur; the only time at which the grade of concentrate fell was during surge periods, when the cells were overloaded. Reagents were fed continuously in both runs in the following amounts per ton of feed: "S" brand sodium silicate, 16.0 lb.; coconut fatty acid, 3.33 lb.; pine oil, 0.33 lb. The pH of the tap water used was 8.5; pH of the ball-mill discharge, 10.2; and of the flotation feed, 8.3. Sampling of concentrate and tailing at various times during the runs gave the information listed in Table 2. The relationship of specific gravity of concentrate to recovery of feed in the concentrate is shown in Fig. 2. Specific gravity of a sample of all cleaned concentrate from run 1 was 4.395. The concentrate was above 98 per cent in $BaSO_4$ content, and recovery of $BaSO_4$ in the concentrate was 90 per cent.

Table 2.—*Records of Sampling*

Sample No.	Run 1		Run 2	
	Percentage Feed as Concentrate	Sp. Gr. of Concentrate	Percentage Feed as Concentrate	Sp. Gr. of Concentrate
1	78.5	4.395	90	4.34
2	76.2	4.395	78	4.37
3	88.9	4.32		4.42

In a third test-plant run the ball mill was omitted. Enough Raymond-milled crude for the whole run was conditioned, at 50 per cent solids, with 12.5 lb. per ton of "S" sodium silicate. The other reagents, 2.75 lb. per ton of coconut fatty acid and 0.2 lb. per ton of pine oil, were added in the usual

manner. In this test a recovery of 78 per cent of the feed in the concentrate was obtained, with a specific gravity of 4.36. Although this result is almost as good as those obtained from the wet-ground feed, it is indicated that the wet grind with the sodium silicate is to be preferred.

This separation is unique in two ways: first, as mentioned, flotation was carried out on a finely ground pulp in the presence of all natural and grinding slime; second, grinding was done in a steel ball mill, with steel balls, and no measures were taken to prevent contamination of the mineral surfaces with iron or iron oxide. Probably the sodium silicate, which acted as a depressant for the small amount of iron oxide in the crude rock, acted to prevent contamination of quartz with iron and its subsequent flotation by the fatty acid. Frequently contamination by iron oxide interferes with flotation separation of nonmetallics by fatty acids, especially when silicates are being treated. Phosphate-rock separation is another case in which iron apparently exerts little influence, and no precautions are taken to prevent interference by this element.

Calcination Treatment of Concentrates

As mentioned, the barite has a distinct gray cast due to intimately distributed carbonaceous matter. The ground crude rock has a reddish hue due to iron oxide. In the flotation operation the latter mineral is almost completely rejected, and the concentrate is gray, with no visible red coloration. The reflectivity of flotation concentrates, as measured by the Hunter Multipurpose Reflectometer, ranged between 55 and 60 per cent and appeared to bear some relationship to purity of concentrate.

It was proposed to increase the reflectivity of the concentrates by calcination to destroy organic matter. It was found that calcination at 1000°C. for 2 hr., in an evaporating dish containing barite to a depth of about 1½ in., burned off all carbonaceous material. The reflectivity of burned concentrates ranged between 70 and 71 per cent. The calcination brought out a

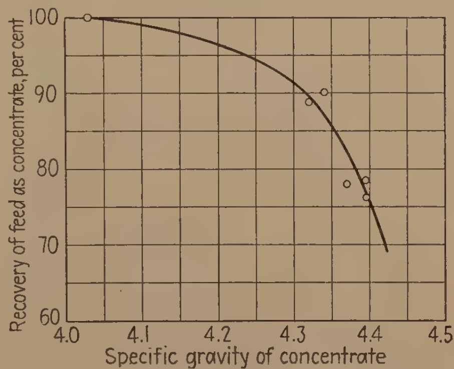

Fig. 2.—Continuous pilot-plant flotation tests.

distinct red color in the concentrates in spite of the fact that they contained only about 0.5 per cent R_2O_3. The iron mineral that caused this red color was apparently in a different form from that rejected by flotation, because unburned concentrates bore no visible red color. To nullify the coloring effect of iron, concentrates were mixed with 0.5 per cent of either NaCl or $BaCl_2$. When calcined, these products were free of reddish hue and gave reflectances of 75 to 78.6 per cent. Aside from the cost, $BaCl_2$ was preferred to NaCl in the calcination because there was somewhat less tendency for the barite to sinter when it was used. In either case, the calcined product would have to be reground, but the material was only loosely bonded and could be reground with little consumption of power.

Flotation concentrate has been found objectionable when used directly as a drilling-mud weight, because the films of fatty-acid compounds repel water and cause entrainment of gas bubbles in the drilling mud. Therefore calcination of the concentrate to destroy the fatty-acid film is now practiced in a plant operating on this deposit in Arkansas.

Summary

Flotation separation of barite shale from Magnet Cove, Ark., was studied. It was found that it was necessary to grind the crude rock to a fine particle size, 100 per cent through 325 mesh, to separate the barite and siliceous gangue. After liberation of the minerals, satisfactory separation of barite and gangue could be obtained by froth flotation without desliming, with sodium silicate, a fatty acid, and pine oil as reagents, but with considerably increased quantities of these reagents. Coconut fatty acid as a collector was somewhat superior to oleic acid.

Test-plant runs showed the process to be more efficient in continuous operation than in laboratory batch machines. About one ton of crude rock was treated in the test plant, and concentrate of 4.395 sp. gr. was produced. Barium sulphate content of the concentrate was above 98 per cent, and a recovery of barium sulphate of approximately 90 per cent was obtained.

The concentrates were discolored by carbonaceous matter, but it was shown that this could be destroyed by calcination. The addition of a small amount of NaCl or $BaCl_2$ was necessary to prevent development of a red color by the calcination.

Acknowledgment

Acknowledgment is made to Dr. Alton Gabriel, associate chemist petrographer, Eastern Experiment Station, Bureau of Mines, for his advice and for microscopic examination of many flotation products.

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DISCUSSION

H. S. Spence,* Ottawa, Ont.—This paper by Norman and Lindsey is a valuable contribution in showing what can be done by a character study and the application of modern flotation methods to an ore that because of its composition and intimately intergrown impurities would until recently have appeared to offer little encouragement for successful beneficiation. That the authors were able to obtain such good results on such comparatively unpromising raw material reflects credit on their ingenuity; and their solving of what must have been a knotty problem will offer distinct encouragement to further work along similar lines with other nonmetallics.

The good results achieved by the combination of an initial Raymond dry grind followed by a wet grind in the preparation of the flotation feed are of interest. The authors do not state why this rather unusual procedure was followed, and it might have been indicated whether it was done with some particular purpose in view or whether they merely used a Raymond-ground product that was furnished to them for the tests.

Especially interesting are the good results achieved without any desliming, and the failure of the steel ball mill to introduce any objectionable iron into the finished product. Possibly the exceedingly fine natural state of subdivision of the silica present in the ore contributed to prevent any material abrasion of the mill steel. A note on the character of this silica, whether it is in fine quartz grains or of an opaline or chalcedonic nature, would have added interest to the paper, as would have a more detailed petrographic description of the ore.

The authors evidently devoted considerable pains to trying to achieve a white product, but they also stress the necessity of calcining barite flotation concentrates for oil-well drilling in order to destroy the objectionable fatty-acid film on the particles; hence, it would be of interest to learn whether the ore from this particular Arkansas deposit is being employed solely for use in drilling mud, and also to know the authors' opinion as to the commercial practicability of processing it to make a white barite acceptable to the trade for general industrial employment by the methods outlined in their paper. Information would also be welcome as to whether the use of NaCl or $BaCl_2$, combined with calcination, is general practice for destroying the off color due to iron

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in barite ores, or whether such treatment might possibly result in objection from the trade. It would seem, from general considerations, that the 98 per cent concentrate made might find a ready market with the lithopone and barium chemicals trades, and therefore that the additional expense of calcination with $BaCl_2$, necessary to overcome its inherent coloration and enable it to compete with naturally white ore, might be avoided.

The almost complete separation of the contaminating red iron oxide mineral present in the ore is a rather surprising and a valuable achievement, and more information on the precise character of this mineral might be furnished, especially since the paper states that iron is present in another form also, which was not removed and which persisted in the concentrates but of which the identity apparently was not determined. The feasibility of removing by flotation the small but strongly coloring content of iron oxide that so often gives a pink to reddish cast to vein barite ores seems worthy of investigation.

L. R. Harrison,* Malvern, Ark.—The flotation plant that is actually operating in the Magnet Cove area is owned and operated by the Magnet Cove Barium Corporation. It first went into operation on June 3, 1940, with a capacity of 50 tons of product per day and has been increased until at the present writing (July 1941) the capacity is 125 tons product per day.

The plant is unique in that it is an all-slime flotation of a nonmetallic mineral. The flotation process used, however, has no relation to that developed by Norman and Lindsey. We have tested the ore by all available reagents and combinations and have definitely established that the process now used is superior to that used by Norman and Lindsey. Their sample ran 4.03 sp. gr., while actually the average would be nearer 3.70 sp. gr. This shows the presence of considerable gangue in the form of clay, silica, and iron oxide and makes the flotation much more difficult.

During the month of June 1941, the plant produced 2000 tons of concentrate with a $BaSO_4$ recovery of 98 per cent plus, made out of average-grade feeds, and floated all minus 325 mesh.

* Manager, Magnet Cove Barium Corporation.

The flotation reagents used in the process cannot be revealed at the present time because of various economic factors and the fact that protection is still in the patent application stage.

O. C. Ralston,* College Park, Md.—A point about this piece of work that justifies emphasis is the fact that flotation separation of the nonmetallic minerals has been hindered by the slime problem. Too frequently the granular material must be deslimed, the slimes discarded and only the granular material treated. This is true in the phosphate industry, the largest of the nonmetallic industries using flotation beneficiation.

The authors of this paper were forced by the nature of the rock to grind to make everything pass through 325 mesh and if they deslimed most of their material would have been discarded. They were forced to develop technique and the proper reagents to treat the whole mass of exceedingly fine material. Due recognition of this accomplishment should therefore be accorded the authors.

J. Norman (author's reply).—Mr. Harrison fails to state in what way the process now being used is "superior to that used by Norman and Lindsey." It was found by petrographic examination of the products of the Norman-Lindsey separation that it was limited only by the ability to grind the flotation feed to liberation of quartz and barite. The separation was adjusted so as to produce as high a grade of concentrate as was possible with high recovery. Mr. Harrison's separation may be more suitable for his purposes, since he mentions a 98 plus per cent recovery of barium sulphate as the only description of his metallurgical results. The Norman-Lindsey process, in the hands of one skilled as well as experienced in the art of froth flotation will accomplish the same or better results, being limited only by the extent of liberation of the component minerals of the flotation feed. If, however, Mr. Harrison has found a way of separating the components of true middlings by flotation process, he should indeed reap rich rewards from the pending patent applications.

Mr. Harrison mentions the specific gravity of the authors' sample, and says that this

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shows the presence of considerable gangue in the form of clay, silica, and iron oxide, which makes the flotation much more difficult. The crude barite used by Norman and Lindsey was not sampled by them, but was representative of material being mined at that time by the Milwhite Company. Also, it should be stated that the presence of additional gangue does not render the separation "much more difficult" except when this occurs as true middling particles. Otherwise only a little knowledge of flotation technique is required to adjust reagents to compensate for the higher proportion of gangue minerals.

Experimental Flotation of Washington Magnesite Ores

By J. B. Clemmer,* Junior Member, H. A. Doerner† and F. D. DeVaney,‡ Members A.I.M.E.

(New York Meeting, February 1940)

Production of magnesium metal in the United States during the past decade has increased from less than 600,000 lb. in 1928 to more than 4,800,000 lb. in 1938.[1] The growing industry has stimulated interest in methods of production and in sources of raw material. All of the present domestic production is from brines. However, the availability of large tonnages of magnesite in the state of Washington, together with the possibility of cheap power, have created interest in the utilization of magnesite for the production of magnesium metal as well as for other purposes.

Magnesite, the carbonate of magnesium ($MgCO_3$), usually is associated with limestone ($CaCO_3$) and dolomite ($MgCO_3 \cdot CaCO_3$). It occurs in crystalline and microcrystalline (sometimes termed amorphous) forms.[2] The more common crystalline mineral is found in various shades of white, gray, yellow and brown, the colors being due to impurities, chiefly iron. Magnesite has a hardness of 3.5 to 4.5 and a specific gravity of 3.0 to 3.1. It forms rhombohedral crystals in the hexagonal system, having a vitreous, pearly luster.

The quarries and mines of the Northwest Magnesite Co. near Chewelah, Wash., are large producers of magnesite for refractories. The deposits are extensive, some veins being hundreds of feet wide. The boundaries of the veins are indefinite, and the magnesite is interfingered with dolomite. Calcite, talc, serpentine, quartz and shale are disseminated through the deposits. The quality of the ore cannot be determined by appearance or by simple tests; chemical analysis is required. These features make it difficult to determine the boundaries or total quantity of workable ore.[3]

Only the higher-grade ores are mined, and a considerable proportion of the magnesite content is discarded by hand-sorting and log-washing methods. After calcining, the sorted product meets the specifications for usual refractory material, as it contains approximately 81 per cent MgO, 7 per cent SiO_2, 6 per cent CaO and 6 per cent $Fe_2O_3 + Al_2O_3$. A sample of the reject material had the following analysis after calcining: 65.4 per cent MgO, 6.3 CaO, 21.1 SiO_2 and 6.5 R_2O_3. The percentage reported as R_2O_3 includes compounds whose chlorides are acid-soluble but give precipitates in ammoniacal solution that are ignited and weighed. The more common constituents are Fe_2O_3, Al_2O_3 and TiO_2.

The sorted magnesite product is not the best grade, even for refractory purposes, and leaves much to be desired for a reduction process. The rejected material contains a large proportion of magnesite. It is apparent that an effective concentration method would yield a higher recovery and a better product. In general, the refractory grade is not suitably pure for other uses. Recent developments in the production of

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[1] References are at the end of the paper.

magnesium metal from magnesite[4] and the trend toward the use of super-refractories[5] probably will create a market for large amounts of high-grade ore.

The present investigation contemplates a primary product containing less than 1 per cent SiO_2, less than 2 per cent CaO and secondary products well within the limits for commercial calcined magnesite.

Flotation of magnesite has been investigated only recently, and the published data are few. Michaelson and Sinkinson[6] floated magnesite from California ores, leaving most of the insoluble constituents in the tailings. Ziebel[7] floated insoluble minerals from Chewelah ore, leaving enriched magnesite in the tailings. In both investigations the analytical data were limited to determination of the amount of acid-soluble and insoluble constituents.

Flotation of Vermont talc-magnesite ores has been developed on a commercial scale.[8] The talc is floated, and the tailings are a marketable grade of magnesite.

Norman and Ralston,[9] in their study of the Chewelah magnesite ores, used both agglomerate tabling and flotation to reject siliceous minerals. They succeeded in obtaining a magnesite product containing less than 3.5 per cent SiO_2, presumably from a deslimed feed.

Doerner and Harris[10] made a detailed study of the effects of many collectors and addition agents on the flotation of low-grade Chewelah ores.

Although the Chewelah ores are relatively free of impurities, the mineral association makes flotation difficult. Direct flotation of the magnesite does not give an acceptable product, because of contamination with talc and serpentine. On the other hand, if the siliceous minerals were floated, the dolomite, calcite and iron-bearing minerals would remain as impurities in the magnesite product. A combination of methods apparently was needed to give acceptable results. In this investigation the following methods were employed:

1. Selectively mined ore, low in calcium and iron, was tested for flotation of siliceous minerals using cationic reagents as collectors in a single-stage procedure.

2. Low-grade ore, high in both impurities, and representing material being discarded in the hand-sorting operation, was tested for flotation of the siliceous minerals with cationic reagents and subsequent "soap" flotation of magnesite, leaving other carbonates as tailings.

Flotation Reagents

Previous papers[9,11–14] have pointed out the utility of the "wetting agent" type of compounds in flotation and have noted that these include compounds that have the long hydrocarbon residue in the positive ion.

Just as the xanthates, Aerofloats, and other collectors for sulphide flotation are thought to ionize with large and multiple-ion anions that become selectively attached to metal cations of sulphide mineral surfaces, so reagents that form large complex cations on dissociation become selectively attached to certain anions, such as silica or silicates. In both instances the attached film imparts new surface characteristics to the mineral particles and makes them separable from uncoated particles by the mechanism of frothing.

The available cationic reagents may be roughly grouped under four classifications:

1. Higher aliphatic amines. Reagents of the classification include dP 243 (du Pont de Nemours & Co., Wilmington, Del.) and AM 1180 (Armour Co., Chicago, Ill.).

2. Quaternary ammonium salts of the higher aliphatic series, such as dP Q (du Pont de Nemours & Co.), and K-1249, 1339, 1340, made by the Emulsol Corporation, 59 E. Madison St., Chicago, Ill.

3. Pyridinium salts, including reagents 660-B, 660-C, S-831, 903, 903-L, 1336, made by the Emulsol Corporation.

4. Alkylated and acylated condensation products of ethanolamines with fatty acids. This classification includes the series DLT

100, 101, 466, 521, 555, 652, 653, 655, 672, 683, 692, 693, 694, 695, 696, 698, 698-B, 699, 907, 1001, 1002, 1003, 1004, 1005, 1006, 1008, 1009, 1041, 1042, made by Dr. W. Kritchevsky, 1401 W. Jackson Blvd., Chicago, Ill.

The Sapamine reagents, manufactured by the Ciba Co., Inc. of New York City and Emulsol reagent 1950-A are not classified.

The present investigation was restricted to the use of cationic collectors DLT 699, dP Q, dP 243, Emulsol 903-L, Emulsol 1950-A, and Sapamine M-S.

Oleic acid was used in the flotation of magnesite with conditioners and frothers as indicated.

Test Procedure

Two groups of samples were obtained from the quarry of the Northwest Magnesite Co. The first group, consisting of high-grade magnesite selectively mined, was represented by a 4-ton sample containing 42.76 per cent MgO, 2.0 per cent CaO, 1.6 per cent R_2O_3, 3.6 per cent SiO_2 and 4.8 per cent insoluble (insoluble in hydrochloric acid; contains in addition to silica, the alumina present in clay). The sample was used for preliminary tests and pilot-plant operation of a single-stage flotation process for the removal of siliceous material using cationic reagents.

The second group consisted of low-grade magnesite, representative of material now being discarded in the mining and sorting operations. Analysis of the samples varied within the limits of 34 to 40 per cent MgO, 5 to 9 per cent CaO, 1.6 to 2.1 per cent R_2O_3, and 6 to 9 per cent insoluble. The samples were used for two-stage flotation, in which the siliceous minerals were floated from the carbonates; then the magnesite was floated from dolomite and iron carbonates.

Flotation of High-grade Ore

Samples of high-grade ore, representing the selectively mined product, were prepared for preliminary tests by dry-crushing in rolls to pass 65 mesh. The material was then wet-ground in a pebble mill for 5 min. to ensure the presence of slimes, to the extent that would be expected in a commercial plant. No ore sample used in flotation tests described in this report was deslimed. Preliminary batch flotation tests were made with charges of 250 grams, floated in a subaeration cell of the M-S type at a pulp density of 27 per cent solids.

Table 1.—*Results of Preliminary Tests with Various Collectors on Wet-ground Washington Magnesite*[a]

Reagent		Magnesite Product		Insoluble Rejected, Per Cent
Name	Lb. per Ton Crude Ore	Weight, Per Cent	Insoluble Assay, Per Cent	
Pine oil	0.16	91.9	2.74	35.2
du Pont 243	0.25	96.2	2.28	35.2
	0.50	92.2	1.68	54.3
	0.75	82.2	1.13	72.0
	1.00	57.1	0.66	88.0
Emulsol 903-L	0.25	79.6	2.53	39.3
	0.50	68.8	2.11	56.2
	0.75	64.1	1.80	65.2
	1.00	57.1	0.80	86.2
du Pont Q	0.25	97.0	3.26	20.0
	0.50	92.9	2.63	38.2
	0.75	88.8	2.08	53.2
	1.00	81.6	0.95	80.4
DLT 699	0.25	93.2	2.39	33.0
	0.50	89.8	1.92	48.3
	0.75	87.8	1.60	57.9
	1.00	85.4	0.90	76.9
Emulsol 1950-A	0.25	82.1	2.82	31.2
	1.00	56.5	1.20	80.0
Sapamine M-S	0.25	93.6	2.40	33.1
	1.00	74.5	1.20	73.2

[a] Pine oil was used as a frother in each test. For each 0.25 lb. per ton of collector, an amount of pine oil equal to 0.16 lb. per ton was added.

The first tests were to compare the merit of various cationic collectors and are summarized in Table 1. Pine oil was used as the frother. Since the tests were for comparative purposes only, the weight recovery of the magnesite and its insoluble content, together with the calculated rejection of

insoluble, was adequate to indicate the trend of flotation.

Reagents DLT 699 and du Pont Q gave the best weight recovery of magnesite in a product containing 1 per cent or less insoluble. Emulsol 903-L and du Pont 243 gave products lower in silica but the recovery of magnesite was low, owing to flotation of much of the fine magnesite. Sapamine M-S and Emulsol 1950-A failed to give an acceptable product. Pine oil, alone, floated the talc, which comprises about 30 per cent of the insoluble in the ore.

The froths were not particularly good. All of the cationic reagents have more or less frothing ability and in the presence of slimes give an undesirably stable, voluminous froth that is difficult to break down. Emulsol 903-L and DLT 699 gave the best froths and were the preferred reagents. A small amount of a modifying agent, such as frother 60, gave a froth that broke down fairly readily. Reagents Q and 243 gave froths unduly stable and difficult to handle.

Comparative tests with and without tannic acid are given in Table 2. The talc was floated with pine oil, then the pulp was conditioned with tannic acid and the insoluble floated, using the collector as shown. Tannic acid gave a magnesite product of lower insoluble and lower lime contents.

TABLE 2.—*Comparative Tests to Determine the Influence of Tannic Acid on Flotation*

Reagents, Lb. per Ton		Magnesite Product, Assay Per Cent			
Tannic Acid	Collector	Wt. Per Cent	MgO	CaO	Insoluble
	dP 243, 0.75	76.3	45.33	1.95	1.38
0.40	dP 243, 0.75	67.5	46.20	1.63	0.76
	dP Q, 0.75	79.4	45.29	1.87	1.20
0.40	dP Q, 0.75	68.2	45.74	1.56	0.67
	DLT 699, 0.75	76.1	45.23	1.93	1.81
0.40	DLT 699, 0.75	74.9	45.82	1.69	1.73

Tannic acid is a mild flocculating agent. It seems likely that one of its important actions is to flocculate the slimes and thereby permit flotation with a minimum of collector. Several inorganic flocculators, such as alum and lime, were tested and appeared beneficial, provided the amount was closely controlled. A slight excess consumes collector and gives poor flotation. Organic flocculants, such as Unifloc and the acid and caustic starches, were found to have merit. Their use for flocculation and clarification of slimes in washery water has already been reported.[15,16]

Comparative flotation batch tests were made with different types of water in which other conditions were identical. Results with distilled water, Rolla, Mo. (comparatively hard), tap water, soda ash-treated water, and zeolite-treated water were identical within the limits of experimental accuracy.

The preliminary tests indicated that acceptable magnesite products could be obtained with dP 243, dP Q, Emulsol 903-L and DLT 699 as collectors.

PILOT-PLANT TESTS

The pilot plant consisted of a grinding and flotation unit and had a capacity of 120 lb. of feed per hour. A 19 by 36-in. rod mill working in closed circuit with a 12-in. spiral classifier constituted the grinding unit. The classifier overflow went direct to the flotation unit, which consisted of a conditioner and six subaeration flotation cells, together with accessory feeders of the constant-head siphon type.

The preliminary tests had indicated that a 65-mesh grind was satisfactory, and an attempt was made to obtain such a grind in the pilot plant. An open-end mill was used and a high circulating load maintained to prevent overgrinding and to hold the amount of slimes at a minimum. The mill was 45 per cent filled with pebbles (275 lb.) ranging from 1 to 3 in in size. Pebbles were used rather than steel balls to prevent contamination by iron. The importance of a granular grind cannot be overemphasized in cationic flotation because the presence

of slimes increases consumption of reagents and lowers recovery. The size of the ground material or the classifier overflow was governed largely by the dilution of pulp in the classifier. Water to the grinding circuit was admitted at the head of the ball mill and was regulated to give a classifier overflow containing about 27 per cent solids.

Screen tests of grinding-circuit products from a typical test, together with operating data, are given in Table 3. Although the

TABLE 3.—*Screen Tests on Grinding-circuit Products*

(Feed rate, 2 lb. per min. Circulating load, 285 per cent. 19 by 36-in. mill loaded with 275 lb. of pebbles; mill speed, 38.2 r.p.m. 12-in. spiral classifier; speed 10 r.p.m.; slope 3¼ in. to 1 ft.)

Percentage of solids: ball-mill discharge, 49.1 per cent; classifier sand, 78 per cent; classifier overflow, 26 per cent

Size, Mesh	Weight, Per Cent				Percentage Removed in Classifier
	Original Feed	Pebble-mill Discharge	Classifier Sand	Classifier Overflow	
8	3.3				
10	15.0				
14	14.4				
20	10.0	5.0	6.9		
28	9.3	3.0	4.1		
35	6.7	3.3	4.5		
48	6.3	5.0	6.5	0.7	3.6
65	6.5	8.4	11.4	2.4	7.4
100	5.8	12.2	10.3	5.1	10.8
150	5.3	16.3	19.4	10.0	16.0
200	4.3	14.7	15.8	12.5	22.1
325	5.9	14.7	12.0	19.6	34.7
−325	7.2	17.4	9.1	49.7	74.2
	100.0	100.0	100.0	100.0	

flotation feed (classifier overflow) contained a large amount of fine material, it was essentially granular and relatively free of slimes, the removal of finished fines in the classifier was not particularly good, and much finished material returned to the grinding circuit.

No attempt was made in the initial pilot-plant tests to clean the rougher concentrates because the quantity of magnesite floated was too small to justify re-treatment. In a commercial plant the rougher froth might be cleaned, but it is doubtful that the gain in recovery would offset the possible loss of grade resulting from the return of the middlings to the circuit.

Stage addition of the collector was practiced, since batch tests showed that this was desirable. Comparative tests showing the percentage of insoluble rejected, using

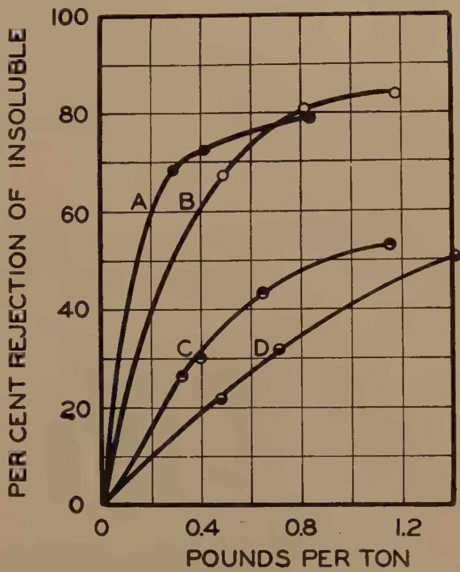

FIG. 1.—FLOTATION OF INSOLUBLE FROM WASHINGTON MAGNESITE.
Using two-stage addition of du Pont 243 (curve *A*), Emulsol 903-L (curve *B*), DLT 699 (curve *C*), and du Pont Q (curve *D*).

two-stage addition of the four cationic collectors, are shown in Fig. 1. In later tests it was found desirable to add the collector in three stages. Less reagent was required, and the froth was easier to control. A series of tests showing the advantage of multiple addition of DLT 699 is given in Fig. 2. Similar results were obtained with 903-L and dP 243.

Results with reagents DLT 699, 903-L, dP 243, and dP Q are tabulated for comparison in Table 4.

Tests with DLT 699.—The first collector tried was DLT 699. Batch tests had indicated that it was slightly less effective for

floating silica than some of the others, but the froth was easier to control. A typical

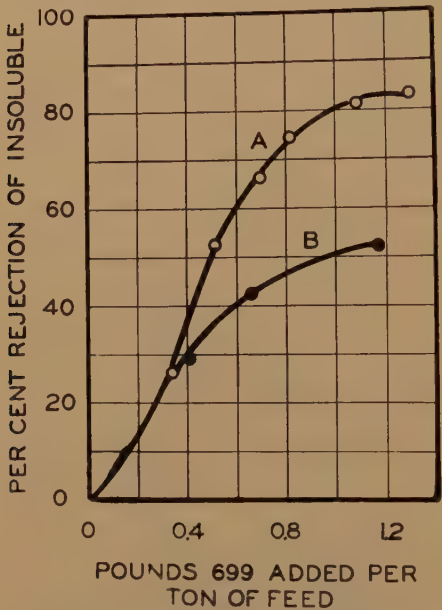

FIG. 2.—FLOTATION OF INSOLUBLE
With three-stage (curve *A*) and two-stage (curve *B*) addition of DLT 699.

test is recorded in Table 5. The distribution as calculated by formula shows that 92 per cent of the silica was rejected and approximately 77 per cent of the magnesite recovered in a product assaying less than 1 per cent SiO_2 and less than 2 per cent CaO. The concentrates and tailings from the individual cells were sampled and assayed to determine where the silica was floating. The analyses, together with the calculated percentage of silica floated in each cell and the distribution of the total silica rejected, are given in Table 6.

The results indicated that the pilot plant was working at full capacity at a feed rate of 120 lb. per hour, and the six roughing cells were necessary to give an acceptable product. Talc was floated in cell 1 with frother 23; then DLT 699 was added to cells 2, 4 and 5 to float the silica. The bulk of the silica floated in cells 2 and 3, but the froths were diluted with slimed magnesite. The addition of DLT 699 to the end cell was necessary to float the granular silica. A curve showing the calculated distribution of total silica floated in each cell is shown in Fig. 3. The cumulative percentage of silica rejected also is given for comparison.

Results listed in Table 4 show that approximately 1 lb. of reagent per ton of feed

TABLE 4.—*Pilot-plant Tests with Different Reagents*

Reagents, Lb. per Ton Crude Ore							Assay, Per Cent						Percentage Recovery of MgO	Percentage Rejection of	
							MgO		CaO		SiO_2				
Tannic Acid	Starch	Frother 23	DLT 699	Em. 903-L	dP 243	dP Q	Froth	Magnesite Product	Froth	Magnesite Product	Froth	Magnesite Product		CaO	SiO_2
		0.24	0.70								5.02	1.62			66.2
		0.32	0.82								4.04	1.38			90.1
	0.62	0.12	0.99				38.92	44.53	2.21	1.91	4.46	0.99	69.6	50.3	91.0
		0.30	1.09				39.37	43.76	2.28	1.82	4.54	0.87	76.9	55.6	92.0
		0.24	1.30				37.31	44.22	2.29	1.84	4.76	0.93	80.3	52.1	90.1
		0 20		0.82			39.21	44.63			6.98	1.22	71.0		80.0
		0.20		0.99			39.02	44.63	2.31	1.84	6.18	1.45	69.6	27.7	80.0
0.36		0.20		1.01			38.95	43.06	2.21	1.83	5.06	0.64	84.4	35.8	94.9
		0.20		1.18			39.15	43.08			6.50	1.08	96.1		83.8
		0.16		1.30			40.28	43.82			6.02	1.04	75.9		85.7
		0.40			0.83		39.02	42.62			7.44	1.28	89.2		78.8
		0.24			0 86		39.43	42 57	2.35	1.84	8.90	1.27	89.0	32.6	76.4
		0.40			0 94		39.59	42.69			6.64	1.02	85.1		85.4
0.37		0.20			0.91		38.41	44.21	2.15	1.82	8.26	1.11	77.5	43.5	81.2
		0.32				1.6	39 95	43 98	2.38	1.95	7.70	1.14	69.3	27.0	77.9
	0.47	0.24				1.7	38.41	43.85	2.32	2.04	6.22	0.74	80.3	40.4	88.7

TABLE 5.—*Typical Pilot-plant Test Using DLT 699 as the Collector*
Feed rate, 2 lb. per minute

Product	Assay, Per Cent				Distribution, Per Cent			
	MgO	CaO	SiO_2	R_2O_3	MgO	CaO	SiO_2	R_2O_3
Flotation feed	42.66	2.05	3.39	1.58	100.0	100.0	100.0	100.0
Froth	39.37	2.28	4.54	2.14	23.1	55.6	92.0	57.3
Magnesite product	43.76	1.82	0.87	1.17	76.9	44.4	8.0	42.7

Reagents Used	Lb. per Ton of Ore	Where Added
Soda ash	0.48	Pebble mill
Frother 23	0.30	Rougher cell 1
DLT 699	0.53	Rougher cell 2
DLT 699	0.34	Rougher cell 4
DLT 699	0.21	Rougher cell 5

is required for acceptable results. In one test caustic starch used as an addition agent did not materially influence flotation.

Tests with Emulsol 903-L.—This reagent was tested because it gave a froth that could be easily handled. The compilation given in Table 4 shows that, as with DLT

TABLE 6.—*Analyses of Cell Concentrates and Tailings from Typical Pilot-plant Test*

Cell	Assay, Per Cent SiO_2			Per Cent Floated SiO_2	Distribution of Total SiO_2 Floated
	Feed	Froth	Magnesite Product		
1	3.39	38.30	2.58	25.6	10.5
2	2.58	3.71	1.76	60.5	24.9
3	1.76	5.03	1.67	76.6	31.6
4	1.67	10.09	1.27	27.4	11.3
5	1.27	16.30	1.02	33.3	8.2
6	1.02		0.87		5.5
					92.0
Over-all	3.39	4.54	0.87	92.0	

699, approximately 1 lb. of reagent was required. Tests were made using tannic acid to improve flotation of silica and lime. One test, as shown, indicates that tannic acid is distinctly advantageous for producing low-silica tailings, but it does not influence flotation of the lime.

Tests with du Pont 243.—Several tests were made using du Pont 243 as the collector. The reagent is an effective silica collector but has a pronounced tendency to float nonmetallic minerals. In the presence

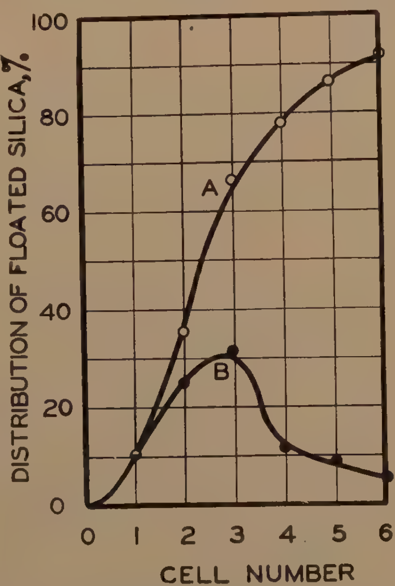

FIG. 3.—FLOTATION OF SILICA IN TYPICAL PILOT-PLANT TEST.
With DLT 699: curve *B*, percentage floated in each cell; curve *A*, cumulative percentage of silica floated.

of slimes it gives an undesirable stable froth that is difficult to handle. A number of the test results are given in Table 4. A test in which tannic acid was used is included for comparison. Batch tests had

indicated that tannic acid was helpful, but it did not prove effective in the pilot plant. Limited ore supplies curtailed extensive testing, and the optimum amount probably was not used.

Tests with du Pont Q.—Only a few tests were made with this reagent. It has more specific collecting properties for silica in the presence of magnesite than du Pont 243, but it is relatively expensive, large amounts are required, and it gives an undesirably voluminous froth. Since the batch tests had indicated that caustic starch improved flotation of silica with reagent Q, this combination was tried. Results, with and without caustic starch, are given in Table 4, and indicate that caustic starch is advantageous in obtaining a low-silica magnesite product.

Flotation of Low-grade Ore

Samples of low-grade ore representing mine rejects were prepared as follows: The crude ore was crushed in rolls to pass 10 mesh. Weighed charges were stage-ground in a porcelain pebble mill to pass 150 mesh and floated in a subaeration cell of 500 or 2000-gram capacity. Distilled water was used in the grind, and the grind water was reserved and used in the flotation tests. The pulp density was about 27 per cent solids.

The use of pine oil as a frother in the flotation of talc caused poor selectivity in subsequent flotation of the magnesite. When the slightly less efficient frother, B-23, was substituted for pine oil and dP 243 to float the insoluble, the subsequent flotation of magnesite with oleic acid was satisfactory. No other frother was required.

Tannic acid and sodium disilicate caused an increase in the ratio of lime to magnesium oxide in the insoluble product and decreased that ratio in the magnesite concentrates. This remarkable reverse, although weak, makes a selective action in each of the two flotation steps and yields a higher degree of separation between the lime and magnesium oxide contents of the ore than was at first suspected. After considerable testing it was found that the best results were obtained by using tannic acid before flotation of the insoluble and then adding sodium disilicate for the second step.

Various possibilities of this method are revealed in the data shown in Table 7. In the test, a 500-gram sample was ground to pass 150 mesh. The pulp was conditioned in the flotation machine with 0.94 lb. of tannic acid per ton of feed for 7 min. and then 0.30 lb. of dP 243 per ton and 0.12 lb. of B-23 per ton were added. The siliceous froth was removed over a period of 15 min. The pulp was then conditioned with sodium disilicate (0.28 lb. per ton) for 7 min.; then with five successive additions of oleic acid (each 0.2 lb. per ton), five separate magnesite concentrates were floated during 3-min. intervals. This procedure gave a siliceous froth, five magnesite concentrates and tailings from the latter having the weights and calculated mineral composition as shown in Table 7. Actually much of the magnesium content of the siliceous froth and tailings products was present as talc or serpentine, so that the percentage of the total magnesite in the composite concentrates is considerably higher than the 70 per cent recorded.

The composition of concentrates and the recovery of magnesium oxide for any fraction of the total weight removed may be estimated from the tabulated data. Similar ore containing less calcium will permit higher recoveries. The composite magnesite concentrate contains only 2.2 per cent of impurities other than dolomite. Petrographic analysis indicated the presence of calcite in the insoluble product but none elsewhere. A count of grains from several concentrates based on indices of refraction indicated that dolomite is present in large enough quantity to account for all the calcium present in those concentrates. It was also noted that in the high-grade

concentrates (those first floated) dolomite was found only in the smaller grains, and in concentrates No. 5 the proportion of dolomite was independent of grain size. X-ray diffraction patterns made from several products showed only magnesite in concentrates No. 1, magnesite and dolomite in concentrates No. 5, and abundant dolomite, a little magnesite, and no calcite in the tailings. The siliceous froth contained calcite, as well as dolomite and magnesite.

The selective flotation of magnesite from dolomite, in this instance at least, is a means of rejecting dolomite. The selectivity is obviously poor, and the smaller dolomite grains float nearly as readily as the largest magnesite grains.

Microscopic examination failed to show the presence of locked grains, but other evidence indicates that most of the minus 150-mesh grains are relatively homogeneous. All the magnesite concentrates looked exceptionally clean, either with or without magnification. On the other hand, only about one-tenth of the particles could be used for measurement of the index of refraction.

TABLE 7.—*Results of Two-stage Flotation on Low-grade Ore*

Product	Weight, Per Cent	Assay, Per Cent				Percentage of Total $MgCO_3$
		$MgCO_3$	$CaCO_3$	Insoluble	R_2O_3	
Heads		74.4	16.6	7.6	1.4	
Siliceous froth	20.1	53.0	17.6	26.9	2.5	14.4
Magnesite concentrate:						
No. 1	7.0	94.5	2.8	1.9	.8	8.9
No. 2	12.3	93.9	3.9	1.3	.9	15.5
No. 3	17.5	90.8	7.2	1.0	1.0	21.4
No. 4	10.5	86.0	11.8	1.1	1.1	12.1
No. 5	12.1	74.2	23.6	1.1	1.1	12.1
Tailings	20.5	55.9	34.8	7.8	1.5	15.6
Composite concentrates	59.4	87.6	10.2	1.2	1.0	70.0
Composite rejects	40.6	55.0	26.1	17.1	1.8	30.0

SUMMARY

Magnesite ore from deposits of the Northwest Magnesite Co. near Chewelah, Wash., divided as selectively mined into high-grade ore and low-grade mine rejects, was tested for concentration by flotation.

Rejection of siliceous minerals from the high-grade ore by flotation to produce higher-grade magnesite concentrates was accomplished in a pilot plant of 120 lb. per hour capacity, using "cationic"-type reagents.

The pilot-plant tests indicated that DLT 699, Emulsol 903-L and du Pont 243 reagents were roughly equal as collectors. Approximately 1 lb. of collector per ton of ore was required, and a product containing approximately 1 per cent SiO_2 was obtained.

Water treatment appeared unnecessary, as similar results were obtained with distilled, soda-ash-treated, zeolite-treated, and a moderately hard tap water.

Three-stage addition of collectors yielded a recovery of 85 per cent of the magnesite in a product containing less than 1 per cent SiO_2 and less than 2 per cent CaO.

Emulsol 903-L and DLT 699 were preferred reagents for silica rejection. Less magnesite was floated, and the froths were easier to handle.

The several groups of cationic reagents as well as the individual reagents of these groups vary considerably in collecting power and selectivity when tested under ideal flotation conditions, but these differences are largely ironed out when siliceous minerals are floated from an essentially nonsiliceous gangue under conditions approaching those of practical operation. The selection of reagents will therefore be made

on the basis of the nature of the froth, behavior toward slimes, and other incidental features rather than on the simple power to float silica.

Caustic starch and tannic acid assisted in flotation of the siliceous minerals.

The low-grade mine rejects, which contained both siliceous minerals and carbonates, such as dolomite and calcite, were treated by a two-stage flotation procedure. First the siliceous minerals and, unexpectedly, calcite were floated, using cationic-type collectors; then the magnesite was floated from dolomite by usual soap-flotation methods.

The results indicated that a broad range of magnesite products, well within specifications for refractory magnesite, could be obtained, and the degree of purity desired with respect to total magnesite recovery could be readily adjustable.

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Sticky-surface Concentration of Gravel-size Minerals

By John Dasher* and James Norman,† Junior Members, and O. C. Ralston,‡ Member A.I.M.E.

(Bethlehem Meeting, October 1942)

Most mineral products are used in the finely divided state, but some are sold in larger sizes. Coal, gravel, metallurgical fluorspar, phosphate rock, hematite, chromite, and other products are sold in sizes coarser than can be treated by many mineral-dressing methods.

Gravity separation of coarser sizes by log washing and jigging is probably as ancient as any process now in use. The performance of jigs on a feed with sufficient difference in density of the minerals is excellent, but many families of minerals are too similar in density to be separated effectively by this method.

The use of heavy suspensions of fine solids to separate coarse coal from bone and slate has been in use for some years. This separation by sink-and-float was limited until recently to light minerals. Development of galena and ferrosilicon as suspension media, by Minerals Beneficiation, Inc., at Mascot, Tenn., Pitcher, Okla., and Crosby, Minn., under patents granted during and since 1938, has permitted excellent results to be obtained on minerals of very similar densities up to 3.3. This process has been put in the hands of American Cyanamid Co. for exploitation, and further research now in progress may increase the density of suspension obtainable so that separations at a density of 4.0 may be practicable before this is published. The Huntington-Heberlein dense-medium process, developed in England, is also being commercialized in the United States by Sink and Float Corporation.

Du Pont has developed a sink-and-float separation using halogenated hydrocarbons as parting liquids; at this writing, the method is used commercially only on coal but has been successful in the pilot plant on many other minerals using tetrabromethane at densities up to 2.95.

While these processes were being developed to extend the range of separation by difference in density to the coarser sizes, the Bureau of Mines was engaged in developing a process that will extend the range of separation by difference in wettability up to 2-in. size. War work has pushed aside its further development indefinitely. The results are published here with the hope that some use may be suggested that will justify post-war development, and so that readers that may be able to make immediate use of the scheme may have the information.

Earlier Work

It has been known since the time of Herodotus (B.C. 484(?)–425) that water-repellent particles will adhere to sticky surfaces. That early historian reports the recovery of gold from the mud of a lake

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by means of feathers daubed in pitch and held in the hands of apocryphal virgins.[1] This semieconomic method would doubtless be successful in recovering fine particles of what may be facetiously termed virgin gold.

Luckenback (U. S. Patent 1370601) passed tailing from gravity concentration through a mat of fiber coated with grease or tar and caught sulphide minerals. Later (U. S. Patent 1448928) he passed flotation-size pulp over a belt, drum or table coated with sticky material like soft gum rubber, rosin, asphalt, pitch or tacky shellac, and more recently (U. S. Patent 1792544) he added as a sticky substance a mixture of candle wax in an oil thinner.

The use in a flotation cell of greasy wood shavings, to which gold particles will adhere, has been proposed.[2] The use of a tar-coated belt around a drum, to recover gold from black sands, has also been patented.[3]

The industrial application of sticky-surface concentration for recovering fine diamonds is described by Gaudin,[2] as follows:

> In the diamond fields, the concentrate obtained from the disintegration of the so-called "blue ground" . . . is passed on a shaking table, the surface of which is smeared with a heavy grease. The diamonds adhere to the grease, and the other minerals do not adhere to the grease, so that after a period of time, it is possible to scrape off the grease with adhering diamonds as a much purer concentrate. Adhesion of the diamonds to grease is no doubt related to their nonpolar structure and to the likewise nonpolar structure of the hydrocarbon molecules of which the grease is made.

The mechanism used for catching the diamonds is essentially the Frue vanner with a canvas belt substituted for the rubber belt; the grease with its load of diamonds is scraped off at the lowest point and a new coating of grease continuously applied.

Ernst Bierbrauer developed an adhesion process for coarse particles.[5] He reports the use of fatty acids or other anionic collectors to film phosphate rock and other soap-floating minerals, which are separated by pressing the particles into a substantially solid pitch with a roll covered with sponge rubber, or reversing the separation by pressing the particles into ice. The water-repellent grains adhered to the pitch, and the wetted grains did not. The separation was reversed with ice. None of his work mentions having the sticky surface immersed in water, although the grains were wet. Our findings are that immersion is essential. Otherwise Bierbrauer has come nearer to a commercial success than prior experimenters.

Batch Experiments

A laboratory model of Bierbrauer's device, "the picking machine," was constructed and tried at the Bureau of Mines Eastern Experiment Station. Only fair separation was obtained. It was difficult to prevent the gangue from being embedded in the pitch if enough pressure was used to make the concentrates stick. Attempts were made to study the effect of varying the belt speed and the consistence of the sticky substance on the belt. Results obtained could not be duplicated and were never satisfactory.

A simple and crude experiment showed two faults of the Bierbrauer process. A greased cloth lowered into a pan of filmed material covered with water had very selective adhesion for the filmed particles and none for the gangue. Two facts were learned from experiments of this type: (1) that the moist particles used in the Bierbrauer process are not wet enough, as contact with the particles under water gives more selective adhesion; and (2) that very small pressures between the sticky surface and the mineral mixture were best

[1] References are at the end of the paper.

employed. With these low pressures, softer sticky but water-repellent substances must be used. With this in mind, the machine was remodeled to submerge the nip of the feed rolls in water but the new arrangement gave only a little improvement in separation. The sticky substance used consisted of 20 to 70 per cent gilsonite with the remainder No. 3 cup grease.

The mechanism of nonadhesion to water-repellent surfaces for water-wetted gangue was investigated. A clean glass plate, representative of unfilmed gangue minerals, was immersed in water and rubbed with different water-repellent substances, with the following results: (1) Contact could not be made with the first class of substances, which included kerosene and oil; (2) contact could, however, be made with substances of medium consistence, such as cup grease; (3) contact could not be made with the third class of sticky materials, which included the occasionally brittle solids such as tar, pitch, and gilsonite.

From these considerations, it would seem best to operate the picking-machine belt with water-repellent sticky surface of the consistence of class 1 or of class 3. However, the class 1 oils are too weak to support the filmed material and hence cannot be used to lift this material away from gangue. The class 3 sticky materials are so hard that it is almost impossible to cause filmed material to adhere to them. This leaves one other possibility—that of using a class 2 sticky water-repellent substance and bringing the mixture to be separated against it with a force small enough so that the water-wetted particles will not be smeared with the sticky material.

It was found that particles would adhere selectively to sticky substances of medium consistence if brought into contact under water with pressures in the order of their own weight. This agreed with the application of sticky-surface concentration of diamonds. The diamond table, however, had two principal disadvantages. It was operated intermittently, and its grease consumption was high. These factors would prevent its application in the beneficiation of low-priced minerals, such as most nonmetallics.

The use of a rotating semisubmerged cylinder with a horizontal axis and with the material fed onto the sticky inside surface was tried, but it was found advisable to change the design of the machine to that of a frustum of a cone with horizontal axis, instead of using an inclined drum. The cone would have a more nearly uniform angle of emergence of the sticky surface from the water. Such a cone 15 in. in diameter at the large end, 6 in. in diameter at the small end, and 18 in. long, provided with a flange 15 in. in diameter, 2 in. from the small end, to allow the cone to rotate on rolls as though it were cylindrical, was constructed. A tank and bearing rolls were also constructed. The tank was filled with water to the axis of the cone. The filmed material was then sprinkled into the small end of the cone, whose inside surface was coated with cup grease, while the cone was rotated and tapped by hand. The rotation was continued until the unfilmed particles that did not adhere had rolled out of the large end of the cone. The adhering particles were removed by placing the cone with its axis vertical, with its small end in a bucket, and washing the adhering particles into the bucket with a strong jet of water.

Experiments with the cone worked satisfactorily and optimum results were obtained if a thin coat of cup grease was used for small particles and a thick coat up to ¼-in. was used for the coarser particles. Particles from 10 mesh up to 2½-in. diameter were thus separated successfully, although particles coarser than 1 in. would not lift above the water surface. However, a properly placed cutter will remove these coarse particles.

Separations were made on limestone-granite mixes, on feldspar quartz, on

kyanite quartz, on barite quartz, and on Tennessee pebble phosphate rocks. The concentration of limestone, feldspar, barite, and phosphate gave almost perfect results, but badly weathered kyanite did not respond properly. A fuel-oil coating on the filmed material did not affect the separation obtained. The granite tailing from the granite-limestone mix was repassed as many as seven times without showing any tendency to adhere to the cup grease, as contrasted with the large adherence on the second pass through the Bierbrauer machine. Standard flotation reagents to apply the initial film were used in all cases. Some trials were made on the fluorite-calcite-quartz separation. Results were not very good. Only 80 per cent recovery of an 87 per cent fluorite concentrate was obtained. Usual flotation reagents (oleic acid and quebracho extract) did not show preference for fluorite. However, when calcite was made to adhere first in an acidified solution and fluorite recovered from the tailing in neutral solution, the results mentioned above were obtained. Although virtually a perfect separation was obtained with cationic reagents between feldspar and quartz, neither of which is filmable with soap, the attempt to separate quartz from phosphate rock with cationic reagents was not successful. The reagents tried were "lupomin" (Jacques Wolf), cetyl trimethyl ammonium bromide (du Pont), and an impure lauryl (Lorol) amine hydrochloride (du Pont). The quartz was made to adhere to the grease but most of the phosphate did also. The failure of this "backward" separation is due perhaps to the high contact angles (about 70°) of calcium phosphate, calcite, and other soap floaters. This high angle indicates that while these minerals are *not* self-floaters and resistent to wetting by water they are *almost* self-floaters and require only a very slight activation to cause them to adhere to the cup grease in preference to the water. This activation is provided by the presence of water-insoluble soaps in the cup grease used.

Continuous Concentrator

Before commercial application is possible, methods of removing the adhering particles and renewing the sticky surface continuously must be developed. Softening the sticky substance at the top of the cone with flame or radiant heat would permit the adhering particles to drop off because of their own weight or to be combed out with a rotating heated comb. Some type of applicator to smooth the surface and replace the lost sticky substance also needs to be developed.

A larger cone with motor drive was constructed to test these methods. A diagrammatic sketch of its action is shown in Fig. 1.

A photograph of this machine was published with an explanatory paragraph in the Annual Report of the Nonmetals Division in November 1941.[6] The cone was 28 in. long, 30-in. diameter at the large end and 12-in. diameter at the small end. It was fitted with two 30-in. diameter flanges, 3 in. from the large end and 4 in. from the small end, fitted with ⅛ by ½-in. steel tracks to roll on 3-in.-diameter flanged wheels. One pair of wheels was driven through a stuffing box by cone-pulley V-belt drive from a gear reducer, and a ⅙-hp. d.c. motor. A wide range of speeds was available. The cone was mounted in a 58 by 40-in. rectangular tank 18 in. deep, set on an angle-iron frame that raised the tank 13 in. from the floor. Two funnels were placed in the tank bottom, to permit removal of concentrate and tailing. The tapping arm was actuated from the gear reducer by a cam lifter opposed by a spring. The particles melted from the grease by the flame dropped into a launder and were washed down and out of the feed (small) end of the cone. Make-up grease was added to the applicator intermittently as needed and overflowed slowly through

the notched lip. Complete removal of the adhering particles with the gas burner used was possible only at slow speeds, such as 5 ft. per min. for ¼-in. particles.

down in melting and reverted to a mixture of oil and wax. Therefore, it was necessary to find a soapless sticky substance that would melt readily and gel readily and

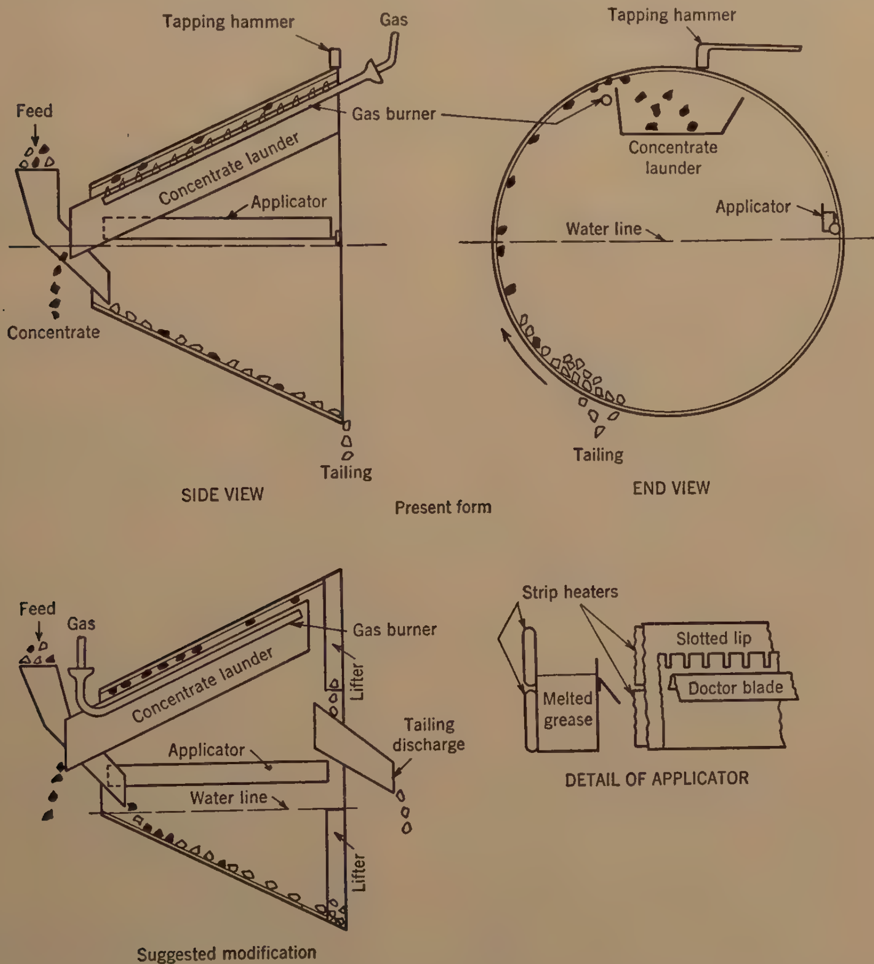

FIG. 1.—DIAGRAMMATIC ACTION OF CONE.

Fig. 1 shows a suggested modification of the machine. Enclosing the large end of the cone with an annular disk and fitting it with lifters to remove water-wetted particles would eliminate the tank, decrease floor space, and eliminate submerged bearings and rollers. This construction, however, is mechanically difficult for small laboratory machines.

It was observed that cup grease broke repeatedly or to devise a method of removing the particles and renewing the surface without heat.

The Shell Development Co. and the Barrett Co. submitted several samples of soapless greases and tars, which they thought might be suitable. One of the Shell products was promising but too thick for use at ordinary temperatures. Additional samples of similar blends were obtained,

of which No. 7 and No. 8 were suitable (A.S.T.M. grease penetration numbers of 140 and 200). Results obtained equaled those obtained with cup grease.

These sticky substances are cheaper than cup grease because they are soap free, and the absence of soap permits "backward" separations, such as quartz from phosphate rock.

The adhering grains pull off some grease; the bigger the piece, the greater the amount of grease. The quantity of grease removed may be decreased by using harder grease up to the point where filmed particles do not adhere well enough from their own weight to be carried up to the top by the revolving cone where they are removed. When the adhering particles are treated with hot water containing alkali, the sticky product is recovered as a scum on top of the tank, and may be reused. The quantities of filming reagents required for the sticky-surface separations are about the same as those required for flotation. The larger particles have less surface but need a thicker film.

Conclusions

To extend the size range of filming processes (flotation and agglomeration), selective adhesion of filmed particles to sticky water-repellent surfaces was studied, and a continuous machine was constructed to make separations of various nonmetallic minerals with normal flotation reagents from 10 mesh to 2½ in. Particles up to 1-in. diameter were lifted out of the water on the greasy inside surface of the frustum of a cone with a horizontal axis.

Bierbrauer's work was checked and found unsatisfactory. The adhesion of filmed particles is only selective when a soft substance is used, and the particles are brought into contact with it under the pressure of their own weight, and under water.

Possible application of the process is limited to rocks that liberate their minerals coarser than 6-mesh, and preferably to those containing minerals used in coarse sizes along with gangue minerals of very nearly the same density, so that separation by the inexpensive heavy-medium process s not practical.

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Magnetic Separation of Sulphide Minerals

By A. M. Gaudin,* Member, and H. Rush Spedden,† Junior Member A.I.M.E.

(New York Meeting, February 1943)

Although the number of minerals that are ferromagnetic‡ or highly paramagnetic is strictly limited, it has been known for some time that many minerals have slight but supposedly characteristic magnetic susceptibilities. With the exception of pyrrhotite, however, sulphides have been thought nonmagnetic.

A new magnetic separator has been made available recently in limited quantity to research laboratories. This device, the Frantz Isodynamic Separator, has been used for the separation of various nonsulphides from each other. Its application to the separation of sulphides, however, does not seem to have been reported.

In the Richards Mineral Dressing Laboratories, the success achieved through the use of this device in separating associated sulphides has been so startling as to lead us to believe that it constitutes a magnificent adjunct to heavy-liquid separation and to quantitative mineragraphy as a tool for the mineral investigator.

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‡ A *paramagnetic* substance is one that, when placed in a magnetic field, becomes magnetized in the same direction as the field, and in proportion to it.

A *diamagnetic* substance, on the contrary, becomes magnetized in a direction opposed to that of the field but in proportion to it.

A *ferromagnetic* substance resembles a paramagnetic substance, except that it is magnetized much more strongly, and not in proportion to the field. For strong fields the susceptibility decreases with increasing field; in other words, saturation takes place.

The Separator

The separator (Fig. 1) consists of:

1. A powerful magnet, the strength of which can be adjusted by means of a rheostat;

2. A vibrating chute of nonmagnetic metal on which the mineral to be separated is made to flow (Fig. 2);

3. A feed bin and adjustable feeder;

4. Receptacles for the collection of products;

5. Means for changing, setting, and accurately reading the two slopes of the chute; viz., the cross slope and longitudinal slope.

The separator is made operative by the special shape of its pole pieces, which ensures a strongly converging field. It is said by the makers of the machine that the pole pieces apply a constant mechanical force to a particle of given susceptibility, regardless of its position in the operating space of the separators.* Susceptible particles, therefore, behave as though of lower specific gravity than under ordinary circumstances. This effect, combined with the slopes of the chute and the action of the vibrator, causes a stratification not unlike that obtainable on a shaking table.

The chute is of rectangular cross section, and shallow; it is broadened toward the discharge end and divided longitudinally by a partition into two compartments that lead to the collecting receptacles.

Although the separator probably can be used to make separations at fairly coarse

* A description of the separator is given in the Appendix, page 574.

sizes, we have used it only in the size range from 35 mesh to 600 mesh and particularly from 100 to 400 mesh.

Ferromagnetic minerals are somewhat troublesome in that they "hang up" in the separator instead of flowing out into the compartment designed to receive the more magnetic fraction. It is desirable, therefore, first to remove metallic iron, magnetite, franklinite, and perhaps pyrrhotite, by means of a hand magnet.

Control of the path taken by any one mineral is obtained by varying the intensity of the current in the separator and the cross slope of the chute. Variations in the longitudinal slope are required essentially in relation to the sizes of the particles, a steeper slope being useful for finer particles.

FIG. 1.—FRANTZ ISODYNAMIC SEPARATOR.

MATERIALS STUDIED

For this study it was deemed desirable to utilize actual mill products, since they represent the type of mineral associations on which the separator could be employed to advantage. A series of mill products was available in the laboratory, collected in 1939–1940 by a number of leading metallurgists. Our thanks go out to them for their hearty cooperation.

The samples were taken wet and shipped to the laboratory under water in glass jars. When received, the sample bottles were sealed with melted paraffin, to keep oxidation to a minimum. Two years later the concentrates and tailings were still in satisfactory condition and showed no visible sign of change during storage.

The samples were wet-and-dry screened, dispersing agents being added if necessary to ensure sparkingly clean grains. Drying of the wet-screened products was done in a low-temperature oven (*ca.* 75°C.) to prevent oxidation and the corresponding possible change in magnetic properties.

Separator Practice

A sample of some 3 to 5 grams of granular material is passed through the separator at very low intensity. The ammeter attached to the magnet allows readings to the nearest 0.02 amp., from 0.00 amp. to about 1.45 amp. The first separation at 0.10 amp. and a cross slope of 10° to 15° gives a preliminary indication as to the presence or absence of ferromagnetic particles. The latter adhere to the upper magnet pole; they must be brushed aside with the magnetizing current off and the vibrator on, by working a few bristles from an ordinary paint brush in the clearance between the chute and the upper pole.

Later, higher intensities, and eventually lower cross slopes, are employed to crudely fractionate the sample into a number of products, which are examined under the binocular microscope to get a general idea of what is being accomplished. From this preliminary fractionation it is then possible to pass to an accurate fractionation producing a limited number of cuts at carefully chosen magnet intensities and chute slopes.

For a quick appraisal of the separation made in the machine, a binocular microscope is helpful; indeed, it might well be said that the separator and the binocular are a team. But the binocular can give only a crude estimate of what the separator is doing; it is worthless in regard to locked particles, and even in regard to the determination of free particles of many minerals.

To assess accurately the Frantz separator, briquettes were prepared of many separated fractions. For convenience in referring from one fraction to another, and for economy in polishing, several small briquettes were mounted together, by a second briquetting operation, to form one large briquette.

All polishing was done on the Graton-Vanderwilt machine and the quantitative mineragraphic technique followed was that described in "Flotation" and in "Principles of Mineral Dressing."

The results of our study are presented under three headings: copper-flotation

Fig. 2.—Vibrating chute.

plants, lead-zinc-flotation plants, silver-flotation plants. These data are necessarily incomplete and extremely condensed. Their main purpose is not only to substantiate the statement that the Frantz separator is particularly useful, but also to outline the qualitative scale of magnetic susceptibility of common sulphide minerals, as well as the

range of variation in susceptibility demonstrated by the various sulphides.

Copper-flotation Plants

Macuchi, Ecuador

At the mill of the Cotopaxi Exploration Co., Macuchi, Ecuador, the principal minerals are chalcopyrite, pyrite, barite, and quartz. The ore contains considerable gold, which occurs largely, if not wholly, as the native metal, the characteristic occurrence being at the chalcopyrite-pyrite grain boundary. Minor constituents are bornite, sphalerite, and chalcocite.

The 200 to 270-mesh fraction of the flotation concentrate was fractionated with a cross slope of 10° and a longitudinal slope of 20° at the following intensities: 0.50 amp., 0.90 amp., and 1.40 amp. This gave four fractions, in order of decreasing magnetic response A, B, C, D. Table 1 shows the composition of the various products, as determined by quantitative mineragraphy.

Table 1.—*Fractionation of the Macuchi Concentrate*
(200 to 270-mesh)

Mineral	Mineralogical Assay of Each Fraction, Per Cent by Weight			
	A	B	C	D
	Magnetic at 0.5 Amp.	Magnetic at 0.5 to 0.9 Amp.	Magnetic at 0.9 to 1.4 Amp.	Nonmagnetic at 1.4 Amp.
Bornite	28.1	2.2	2.1	0.4
Chalcocite	3.0	0.2	1.6	0.7
Chalcopyrite	40.9	92.5	55.1	7.2
Sphalerite	4.8	1.3	5.2	4.4
Pyrite	6.5	2.7	26.1	81.2
Gangue	16.7	1.2	9.9	6.1

In Table 2, the data of Table 1 are rearranged and expressed in terms of the free or locked character of the various particles. This table brings out that there is no free pyrite in fractions A, B or C, no free chalcopyrite in fraction D, and virtually no free particles of any kind in fraction C.

Gold was seen only in sections of products C and D, but the number of particles counted was too small to provide statistical grounds for estimating its behavior in the separator. Since no free bornite, chalcocite, or sphalerite was seen in any product, and

Table 2.—*Fractionation of Macuchi Concentrate*

Mineral	Mineralogical Assay of Each Fraction, Per Cent by Weight			
	A	B	C	D
	Magnetic at 0.5 Amp.	Magnetic at 0.5 to 0.9 Amp.	Magnetic at 0.9 to 1.4 Amp.	Nonmagnetic at 1.4 Amp.
Free particles:				
Bornite	0	0	0	0
Chalcocite	0	0	0	0
Chalcopyrite	13.6	68.6	0.5	0
Sphalerite	0	0	0	0
Pyrite	0	0	0	32.9
Gangue	3.6	0	0.4	0.4
Locked particles	82.8	31.4	99.1	66.7

since "gangue" is a collective term, these data provide little information concerning these minerals. On the contrary, a clear-cut separation of chalcopyrite was obtained, with a tendency for bornite to respond to a weaker field than calcopyrite.

New Cornelia, Arizona

The 270 to 400-mesh fraction of the concentrate from the New Cornelia mill was fractionated at 0.6, at 0.85, and at 1.4 amp., and the fractions were briquetted and counted with the results shown in Table 3.

Table 3.—*Fractionation of New Cornelia Concentrate*
(270 to 400-mesh)

Mineral	Mineralogical Assay of Each Fraction, Per Cent by Weight			
	A	B	C	D
	Most Magnetic			Least Magnetic
Bornite	17.9	9.3	1.0	0.1
Chalcopyrite	61.0	75.5	18.1	3.5
Pyrite	2.9	4.2	34.9	53.3
Gray sulphides[a]	8.0	0.8	3.2	2.2
Gangue	10.2	10.2	42.8	40.9

[a] Chalcocite, tetrahedrite, tennantite, enargite, molybdenite, sphalerite.

Table 3 shows that bornite is more susceptible than chalcopyrite, and the latter than pyrite, thus confirming the observations made on the Macuchi products. The gray sulphides were in small quantity and largely locked; their identification was not attempted. However, it seemed as though tennantite-tetrahedrite were present, particularly in A, sphalerite in C and D, and molybdenite in D.

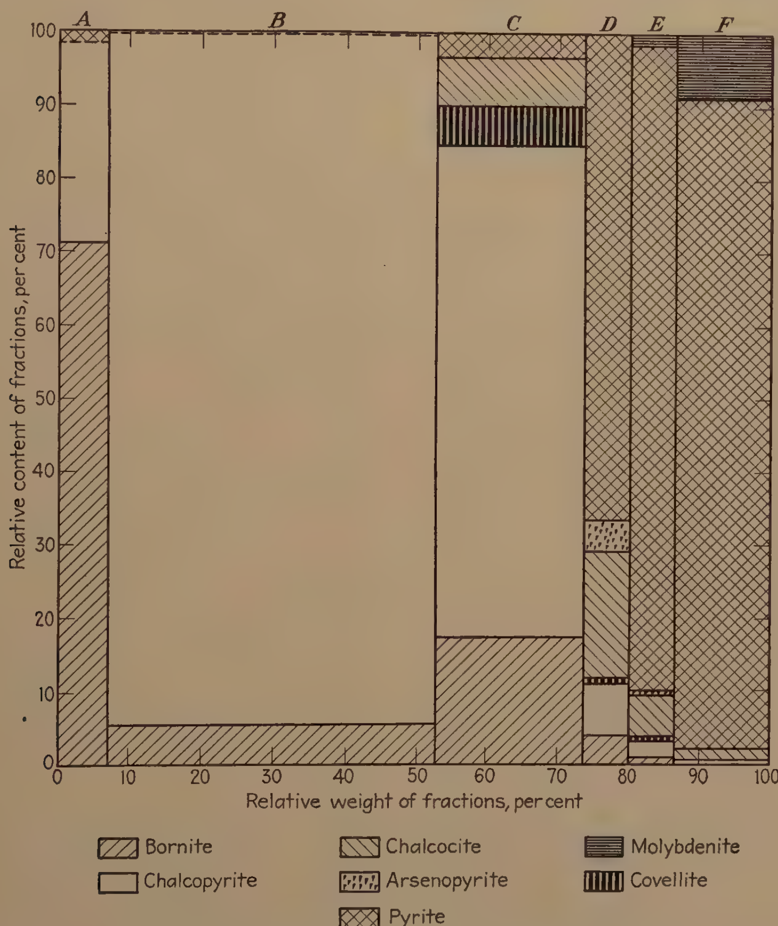


FIG. 3.—FRACTIONATION OF UTAH COPPER CONCENTRATE.

Arthur, Utah

The 150 to 200-mesh fraction of the Utah Copper concentrate was first fractionated with tetrabromethane (sp. gr. 2.96) to remove the gangue. The sediment was fractionated magnetically at 0.77 amp., 0.92 amp. and 1.0 amp., all with a cross slope of 13°, and at 1.4 amp. with cross slopes of 3° and of 30′. Close control of the amperage was made necessary by the large quantity of chalcopyrite and its evidently consistent response in the range of a few hundredths of an ampere on either side of 0.85 ampere.

The results obtained confirm that bornite is more susceptible than chalcopyrite, and the latter than pyrite, but they add further

information, as may be seen in Fig. 3. This chart shows clearly that chalcocite is slightly, if faintly, magnetic; that covellite is slightly more magnetic than chalcocite, and that molybdenite is less susceptible than pyrite. We believe that molybdenite may even be diamagnetic. Furthermore, arsenopyrite can be spotted definitely in only two of the fractions, D and E.

Anaconda, Montana

The greater complexity of the Anaconda concentrate and the more intimate intergrowth of the minerals did not prevent a fairly clean-cut separation.

The fractionation was made at 0.77, 0.83, 0.93, and 1.05 amp., with cross slope of 12° and longitudinal slope of 25°, and at 1.40 amp. with cross slopes of 8° and of 2°. The results are presented in Table 4. This brings out the relative positions of certain additional minerals, notably tennantite-tetrahedrite and enargite. The first of these is distinctly susceptible, but the latter, like pyrite, is either not paramagnetic at all or very faintly so.

TABLE 4.—*Fractionation of Anaconda Concentrate*
(200 TO 270-MESH)

Mineral	Mineralogical Assay of Each Fraction, Per Cent by Weight. Fractions Arranged in Order of Decreasing Magnetic Susceptibility						
	A	B	C	D	E	F	G
Bornite	85.6	40.6	7.7	12.8	14.4	4.3	0.9
Tennantite	3.9	20.3	8.7	19.0	9.5	6.1	0.7
Chalcopyrite	6.8	34.1	78.1	55.6	19.7	6.7	0.5
Chalcocite	2.1	1.8	1.1	4.1	16.8	18.2	6.1
Enargite	0.1	2.0	2.2	3.9	6.7	5.0	4.3
Sphalerite	0.1	0.4	0.1	0.0	2.2	0.6	3.7
Pyrite	0.9	0.5	2.1	4.5	30.2	55.4	80.2

It was observed in the course of the microscope counts that the tennantite-tetrahedrite is not of uniform composition, but that the grains more susceptible to filming fast by selective iridescent filming* are more likely to be paramagnetic than those filming slowly. It was also observed that certain grains prone to look now like bornite, then like chalcocite, are really colloidal dispersions of one of these phases in the other, and that the separator sorts out dispersions high in bornite from those low in bornite.

* A. M. Gaudin: Identification of Sulphide Minerals by Selective Iridescent Filming. A.I.M.E. *Tech. Pub.* 912 (*Min. Tech.*, March 1938).

Since well over three-fourths of the concentrate particles in the size range studied are locked, it is not surprising that the separation is not perfect. We feel that it is about as good as could be expected. This is reflected in the average composition of locked particles. As an example, Table 5 gives the average bornite content of chalcopyrite-bornite and pyrite-bornite locked particles.

TABLE 5.—*Percentage of Bornite Content in Two Types of Locked Particles in Fractions from Anaconda Concentrate*

Mineral	Fractions Arranged in Order of Decreasing Magnetic Susceptibility						
	A	B	C	D	E	F	G
Chalcopyrite-bornite	84	34	12	[a]	[a]	[a]	[a]
Pyrite-bornite	[a]	[a]	50	35	25	11	7

[a] The number of grains of this type was insufficient for statistical analysis.

LEAD-ZINC FLOTATION PLANTS

Midvale, Utah

Perhaps because the samples were taken at an unusual time, the Midvale lead concentrate was found to contain an exceptionally large quantity of copper as chalcopyrite. To an extent, therefore, the data on this plant provide a measure of the behavior of minerals from a lead-zinc-copper ore.

A portion of 270 to 400-mesh lead concentrate was fractionated at 1.4 amp. with a longitudinal slope of 20° and a cross slope of 10°, giving two products that are essentially a lead concentrate and a copper-zinc

concentrate (Table 6). Detail fractionation of the copper-zinc product was carried out qualitatively but not quantitatively. The chalcopyrite seemed to concentrate in the range provided by magnet amperage of 0.85 to 0.90, with sphalerite occurring abundantly in both the more magnetic and the less magnetic fractions.

TABLE 6.—*Fractionation of Midvale Lead Concentrate*
(270 TO 400-MESH)

Mineral	Mineralogical Assay, Per Cent	
	Magnetic Fraction	Nonmagnetic Fraction
Galena	8.5	69.2
Sphalerite	32.6	3.9
Chalcopyrite	42.9	1.2
Tetrahedrite and other gray coppers	7.7	1.7
Bornite, chalcocite, covellite	0.5	0.0
Pyrite	5.2	20.3
Gangue	2.6	3.7

When allowance is made for locked particles, the separation is all the more striking, the magnetic fraction consisting mostly of free chalcopyrite and tetrahedrite and of locked particles and the nonmagnetic fraction mostly of free galena and free pyrite.

In this case it may be said that the separator has clearly sorted sphalerite-bearing locked particles, tetrahedrite, and chalcopyrite on the one hand from galena and pyrite on the other. Actually, if such a separation could be carried out on a practical scale, it might have commercial value, not only because of removal of zinc from the lead concentrate, but also because of segregation of copper from lead minerals. Chemical analyses for lead, copper, and zinc are as follows: in the magnetic fraction, 7.4 per cent Pb, 18.8 Cu, 21.2 Zn; and in the nonmagnetic fraction 60 per cent Pb, 1.23 Cu, 2.5 Zn. The lead-copper and lead-zinc selectivity indices are 11.2 and 8.2, respectively.

In the Midvale products the sphalerite is largely contaminated by inclusions of chalcopyrite. These inclusions, although in minor volume (about one-twentieth of the volume of the sphalerite), might conceivably be the cause of the magnetic response exhibited by the sphalerite.

In order to obtain a more accurate appraisal of the influence of the chalcopyrite inclusions on the magnetic susceptibility of the sphalerite, the zinc concentrate was fractionated at 0.65, 0.8, 1.0, and 1.4 amp., with slopes of 10° and 25°. Table 7 presents the composition of the various fractions as determined microscopically. The startling fact is that there is so little difference between the fractions. Except for the occurrences of a few grains of free chalcopyrite in fraction C and for the occurrence of appreciably more galena and pyrite in fractions D and E, the fractions are not distinguishable. This leads to the thought that the differences in susceptibility of the sphalerite particles are not due to their chalcopyrite inclusions, but to some other reason. Actually, when the particles are viewed with polarized light, crossed nicols, and an oil-immersion objective of medium power (8 or 16 mm.), the internal reflections are deep red or orange for the sphalerite from A, but pale yellow to white for that from E. This suggests that the differences in magnetic susceptibility may be related to the color in transmitted light. Naked-eye comparison of heaps of A and E particles confirms this correlation. Generally speaking, the sphalerite is chocolate colored, but the intensity of the coloration decreases from A to E. Since there is no corresponding decrease in either the quantity of inclusions or the dispersion of the inclusions, the brown color cannot be caused by the inclusions. Actually, similar inclusions in the sphalerite from the Morning mill, whenever present, give it a terne grayish luster, rather than the familiar resinous or honeyed luster of blende.

It is tempting to believe that the color and magnetic variations are due to an iron content in solid solution, and no doubt this can be ascertained if desired, but because of the pyrite content of fraction E, analytical verification would require much painstaking microscopy, supplemented by heavy-liquid fractionation.

TABLE 7.—*Fractionation of Midvale Zinc Concentrate*
(270 TO 400-MESH)

Minerals	Mineralogical Assay of Each Fraction, Per Cent by Weight				
	A Most Magnetic	B	C	D	E Non-magnetic
Sphalerite	98.2	98.6	96.7	88.8	70.6
Chalcopyrite	1.4	1.0	2.1	2.3	1.3
Pyrite	0.1		0.7	4.3	16.7
Galena	0.2		0.5	4.5	11.4
Miscellaneous	0.1	0.4		0.1	

Morning, Idaho

The zinc concentrate from the Morning mill, size 150 to 200-mesh, was first washed in acetone, to remove adhering slime particles. Then the sparklingly clear particles were fractionated at 0.32 amp., with cross slope of 15° and longitudinal slope of 25°; at 0.52 amp. and 1.0 amp. with the same slopes, and at 1.4 amp., with slopes of 6° and 25°. The five fractions were then viewed under the binocular microscope, with the Ultropak microscope, and in polished section.

The binocular indicated the presence of two kinds of sphalerite occurring *simultaneously in each cut*, one black and opaque and the other translucent in various shades of brown, ranging from medium brown in B to pale yellow in E. In addition, much siderite was seen in A; it is easily revealed by immersion in an oil of suitable index of refraction (e.g., methylene iodide). The Ultropak makes the distinction between sphalerite and siderite very clear; the latter is clear to very pale yellow, while the first glows with fiery internal reflections.

The two kinds of sphalerite seen under the binocular are probably particles devoid of inclusions (translucent) and peppered by inclusions (opaque), respectively. The inclusions are commonly galena, pyrite and chalcopyrite; they occur frequently in tremendously fine state of dispersion, so fine, indeed, as to challenge identification of the dispersed phase.

Table 8 shows that, in spite of the intimate locking, the magnet first removes the siderite, then the sphalerite.

TABLE 8.—*Fractionation of Morning Zinc Concentrate*
(150 TO 200-MESH)

Mineral	Mineralogical Assay of Each Fraction, Per Cent by Weight				
	A Most Magnetic	B	C	D	E Non-magnetic
Siderite	47.0	9.6	0.7	0.0	0.0
Sphalerite	44.0	81.5	96.2	84.7	58.8
Galena	6.2	5.9	1.4	4.7	5.4
Pyrite	0.7	1.5	0.8	6.6	33.1
Silicates	1.7	1.1	0.8	1.9	2.3
Others	0.4	0.4	0.1	2.1	0.4

The pyrite is clearly concentrated in fraction E. Galena presents a curious case of a mineral concentrated in fractions A, B, and E, not because it is sometimes magnetic and sometimes nonmagnetic but because most of it is locked, some being attached

TABLE 9.—*Percentage Distribution of Galena in Morning Zinc Fractions*

Galena Attached to	Fraction				
	A	B	C	D	E
Siderite	30	29	0	0	0
Sphalerite	5	13	66	30	42
Pyrite	0	0	0	3	7
One of several others	0	0	0	30	14
Several minerals	65	58	34	37	37
Total	100	100	100	100	100

to siderite (goes to A and B), some to pyrite (goes to E), some to sphalerite (goes to C, D, and E), and some to several of these minerals together (Table 9).

Sullivan, British Columbia

Two sizes of the lead concentrate from Sullivan, B. C., were fractionated, the 200 to 400-mesh size and the size retained in the wet state by a 400-mesh sieve but passing it after drying.

The 200 to 400-mesh size contained some white mica, which, because of its shape and peculiar magnetic response, caused difficulty in the separator. Accordingly, the mica was removed by settling the sample in tetrabromethane (sp. gr. 2.94). The sediment was washed, dried, and passed over the separator at 0.2 amp., 0.5 amp. and 1.4 amp., with a cross slope of 12° and a longitudinal slope of 20°.

The finer size contained very little mica and was separated directly at 0.5 amp. and at 1.4 amp., with slopes at 12° and 20°. Passage through the separator was disappointingly slow in this fine size.

Table 10 shows great similarity in the results obtained on the two sizes, thus confirming the qualitative observation that while close sizing ahead of separation is desirable, it is not entirely necessary.

The separation of the pyrrhotite from the marmatite and the galena is just what would have been expected. Actually, products A and B consist almost wholly of locked particles (they each contain about 10 per cent assorted free particles and 90 per cent locked), while C contains 54 per cent free galena and D, 95 per cent free galena. If magnetic separation could be made cheap enough to become a practical tool in the size range and susceptibility range of interest in connection with sulphides, it would provide a means to segregate for regrinding locked particles not segregable by flotation. But such a development is, of course, not yet at hand.

TABLE 10.—*Fractionation of Sullivan Lead Concentrate*

Mineral	Mineralogical Assay of Each Fraction, Per Cent by Weight						
	A Most Magnetic	B	C	D Least Magnetic	E	F	G
	Size 200 to 400-mesh				Size 400 Wet to 400 Dry Mesh		
Pyrrhotite	58.0	17.0	1.0	0.0	64.5	13.8	0.3
Marmatite	8.5	32.6	15.4	0.6	4.8	30.8	2.9
Galena	27.0	41.6	76.9	97.0	27.4	50.9	95.4
Chalcopyrite	4.0	7.6	1.7	0.0	1.9	2.4	0.2
Other minerals[a]	2.5	1.2	5.0	2.4	1.4	2.1	1.2

[a] Largely pyrite, arsenopyrite, jamesonite, and tetrahedrite; silicate gangue is inconsequential.

Tooele, Utah

The Tooele sphalerite is much paler in color than that from either Midvale or Sullivan. Nearly pure fractions are readily

TABLE 11.—*Fractionation of Tooele Zinc Concentrate*
(150 TO 200-MESH)

Mineral	Mineralogical Assay of Each Fraction, Per Cent by Weight				
	A Most Magnetic	B	C	D	E Least Magnetic
Sphalerite	98.1	98.5	99.0	98.9	81.5
Chalcopyrite	1.3	0.3	0.3	0.2	0.5
Pyrite	0.1	0.2	0.2	0.2	11.1
Galena		0.5	0.1	0.2	3.0
Miscellaneous	0.5	0.5	0.4	0.5	3.9

obtainable by magnetic fractionation at 0.95 amp., 1.1 amp., 1.4 amp., with slopes of 13° and 25°, and at 1.4 amp. with slopes of 6° and 20°. Table 11 presents the quantitative mineragraphic results obtained. Although the various fractions differ in appearance to the naked eye, there is little difference in zinc content until fraction E is

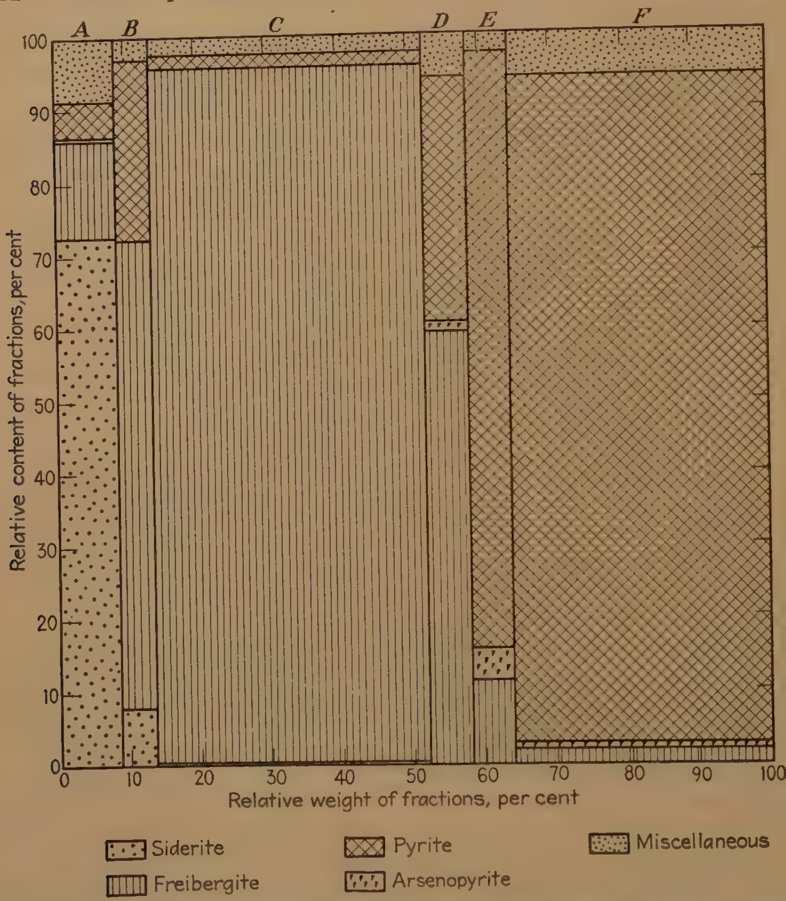


FIG. 4.—QUANTITATIVE MINERAGRAPHIC RESULTS IN FRACTIONATION OF SUNSHINE CONCENTRATE.

TABLE 12.—*Color of the Tooele Sphalerite as Seen in the Ultropak*[a]

Color	Percentage of Particles Viewed in Each Fraction				
	A	B	C	D	E
Red brown	31	2	0	0	0
Orange	34	16	5	0	0
Yellow	33	76	48	21	10
Pale yellow	2	5	40	62	36
Clear	0	1	7	17	54

[a] This table presents the average color perceptions of two observers.

considered. The zinc mineral must vary in composition, since it varies in both optical and magnetic properties. Table 12 shows that the variation in color is very marked.

The data from the various plants making zinc concentrates—namely, Midvale, Morning, Sullivan and Tooele—show that the sphalerite does not have a uniform susceptibility, and that this lack of uniformity is not due primarily to the occlusion of paramagnetic minerals in the

sphalerite. In fact, the occlusion of chalcopyrite, which is so common, does not account for the observed behavior of the content. A correlation between magnetic susceptibility and color can be noted, but the agreement is imperfect; this suggests

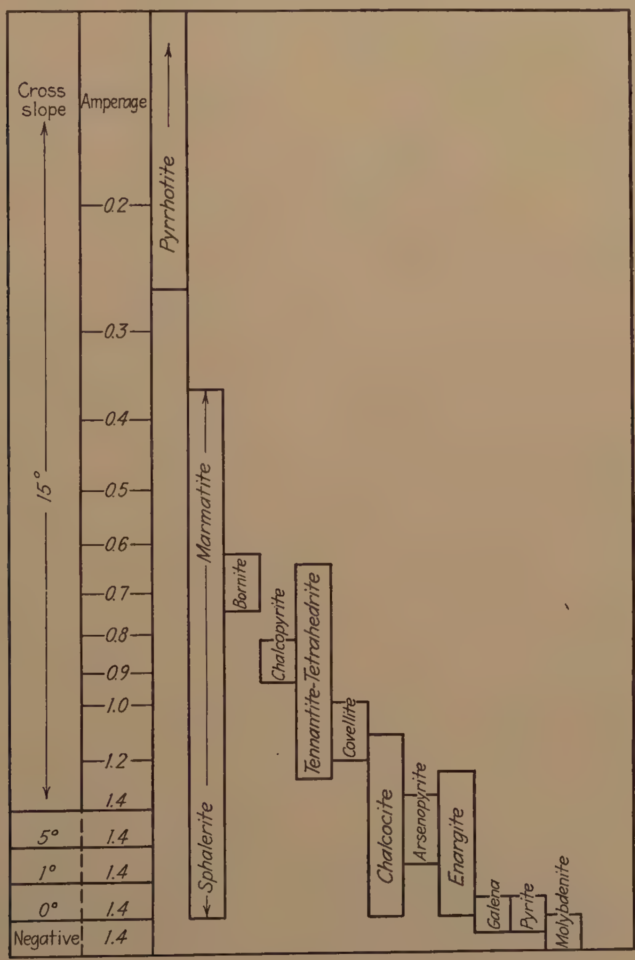


FIG. 5.—MAGNETIC RESPONSE OF SULPHIDE MINERALS.

mineral. It is our opinion that the magnetic susceptibility of the sphalerite is due to iron in solid solution, and that the brown color is caused by foreign atoms in solid solution, particularly iron, copper, and manganese. The grayish color of some otherwise pale sphalerites is due to inclusions, and the dull black color to inclusions occurring jointly with high solid-solution that the coloring ingredient and the magnetizing ingredient are not exactly concordant.

SILVER-FLOTATION PLANT

Sunshine, Idaho

The Sunshine plant was selected because it features a highly argentiferous tetrahedrite (usually called freibergite). Actu-

ally, this mineral and pyrite form the bulk of the concentrate with galena, chalcopyrite, arsenopyrite and ruby silver as minor constituents.

Fractionation of the concentrate was made at 0.5 amp., 0.79 amp., 1.0 amp., and 1.4 amp., with cross slope of 15° and longitudinal slope of 25°, and at 1.4 amp. with slopes of 5° and 25°. Quantitative mineragraphic results are summarized in Fig. 4.

Clearly freibergite is magnetically susceptible to much the same extent as chalcopyrite.

Discussion and Summary

The results presented in this paper give some measure of the magnetic susceptibility of a number of sulphide minerals. These minerals fall into three groups.

The minerals of one group, pyrite, galena, chalcopyrite, and bornite, have been found to display consistently the same susceptibility. Of these four, galena and pyrite are wholly nonsusceptible, but the other two are susceptible.

The second group consists of four sulphides, which we believe have definite, if slight, susceptibilities. They are arsenopyrite, covellite, enargite, and molybdenite. Of these we believe that molybdenite is diamagnetic, and that enargite, arsenopyrite, and covellite are faintly paramagnetic. The evidence on these minerals, however, is not yet as full as might be wished.

The third group, the most interesting, consists of chalcocite, sphalerite-marmatite, pyrrhotite and tennantite-tetrahedrite-freibergite. The minerals of this group all display paramagnetism, even perhaps ferromagnetism, as in the case of pyrrhotite. But the susceptibility varies from particle to particle within the same ore, as well as from ore to ore.

For sphalerite, it has been shown that the magnetic property and the color are largely parallel; also that the susceptibility is not primarily due to inclusions of chalcopyrite. For tetrahedrite-tennantite, it has been shown that some correlation exists between magnetic susceptibility and iridescent filming. It is believed that the variability in the susceptibility is due to variations in composition. The minerals in this group are all notable for their tendency to form solid solutions; it would be but natural for these solid solutions to have unequal susceptibilities.

Fig. 5 summarizes our observations. The results presented in this paper also point out the great possibilities of the Frantz isodynamic separator as a research tool

Acknowledgments

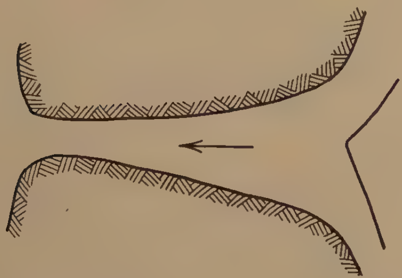
In concluding, we wish to express our appreciation to our associate, Dr. R. Schuhmann, Jr., for his helpful suggestions and to Messrs. S. C. Sun, Risto Hukki, and Frank Bowdish for their assistance.

APPENDIX

Concerning the construction and operation of the separator, Mr. Samuel G. Frantz, President, S. G. Frantz Co., writes as follows:

The Frantz Isodynamic Separator consists essentially of an electromagnet having two long pole pieces shaped to a special contour with a long, narrow air gap between them through which the material is continuously fed. The separating force is effective in the air gap, and a divider or interceptor directs the products into two separate containers. The laboratory separator, Model L1, weighs about 200 lb. and occupies a space about 16 by 21 in. by 20 in. high. It is equipped with hopper feed, chute and vibrator feed, ammeter, and rheostat. The current required for the magnet is about 1.6 amp. maximum at 115 volts direct current. The vibrator uses 60 cycles alternating current at 115 volts. A tube rectifier furnishes direct current for the magnet if direct current is not otherwise available. The pole pieces are 10 in. long and the gap width is about 3/16 in. at the narrowest place. The material to be separated travels parallel to the length of the pole pieces; the magnetic force urges the grain to the left

in the sketch, which shows a cross section normal to the length of the poles:



The magnetic system is carried on a universal mounting, so that it can be oriented in any direction with respect to gravity. One method of use is to set the pole pieces vertical and drop the grain stream straight through the gap from a small hopper at the top about over the base of the arrow in the sketch. The more magnetic particles are urged toward the left where the air gap is a minimum, and are intercepted by an adjustable dividing edge at the bottom of the pole pieces, which directs the two fractions into separate containers.

The separator may also be used with a vibrating feed chute 5 or 10° with the grains fed slowly down over it. This method is particularly suited to the separation of small samples of finely ground materials. The laboratory machine is regularly used with inclined chute for samples of a gram or less. When used in the vertical position on heavy minerals, it has a maximum capacity of 40 to 60 lb. per hour. It is possible to make diamagnetic separations, that is, to separate diamagnetic substances like quartz or zircon having negative mass susceptibilities of the order of -0.3×10^{-6} from substances that are less strongly diamagnetic or are paramagnetic.

The selectivity of the isodynamic separator is due to the shape of the pole pieces, which gives a constant force on a particle of given susceptibility regardless of its position in the operating space; that is, the space in the magnetic field where magnetic forces are effective to cause separation. In other separators, such as the pick-up type, the force varies enormously with the distance from the surface of the pole piece, so that the separation of a given particle will depend not alone on its susceptibility but on the path along which it happens to be fed, and also even on its size.

The reason is that in other separators magnetic forces are exerted on the magnetic particles in the grain mixture by means of localized concentrations of magnetic flux. Typical of such arrangements are the field between a pointed and a flat pole piece, and the field near a magnetized serrated or laminated rotor, and it is inherent in these designs that the operating space is very restricted in volume. For example, in a rotor-type separator, the operating space is a small volume immediately surrounding a line parallel to the rotor axis and passing through the point where the grain stream leaves the rotor. In the old Edison separator, the grain is dropped through a columnar space in which the operating spaces are small regions adjacent to the pole pieces of the magnets, and the rest of the columnar space is not usefully employed for separation. The isodynamic separator has a long operating space, measured parallel to the flow of the grain, compared with other types, and the magnetic field has a special configuration. This gives a long time of action on the particles instead of a short impulse, and the direction of motion assumed is substantially the direction of the resultant of the magnetic and gravitational forces. Because of the special configuration of the pole pieces, the ratio of magnetic force to gravity is substantially constant for any particle, that is independent of its position or path, and the resulting direction of motion is a function only of the mass susceptibility, the initial velocity, and interference between particles. The initial velocity is either made negligible or carefully controlled by properly designed feed nozzles. Interference between particles is limited by feeding a rather thin grain stream, which for maximum flow is in the form of a flat ribbon or curtain of falling grains.

When used with vertical pole pieces and free fall of grain, the isodynamic separator gives good results on free-flowing granular materials of size between about 30 and 100 mesh. Below 100 mesh capacity and efficiency begin to suffer because of grain interference and cohesive forces between particles. Above 30 mesh it is not feasible to feed a good straight stream except through large, low-power gaps. In the laboratory, samples of fines of 200-mesh size or smaller are successfully separated at very slow rates by using inclined pole pieces and the vibrating chute. In every case highly magnetic particles such as magnetite must be removed by a preliminary pass at very low power, otherwise they would stick to the pole pieces. A typical separation involves several passes at increasing intensities.

The Conductance Electrostatic Separator

By Foster Fraas*

(New York Meeting, February 1942)

Most commercial electrostatic separators utilize the electrical property of conductivity, but although based on the same principles, they are constructed in a variety

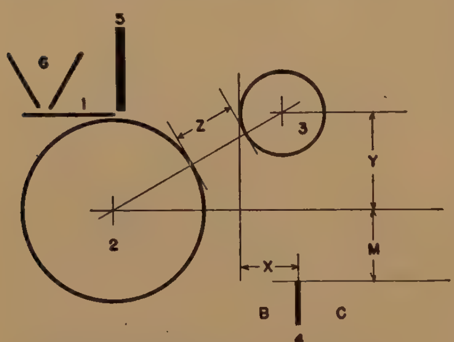


Fig. 1.—Simple roll-type separator.
1. Vibrating feed plate.
2. Cylindrical knurled nickel-plated roll at ground potential. Diameter = 11.4 cm.
3. Cylindrical insulated brass electrode at electrical potential different than ground potential. Diameter = 5.0 cm.
4. Dividing edge or partition of compartmented box. For dividing edge, $M = 7.2$ cm.; for compartmented box, 5.9 cm.
5. Bakelite shield backed with metal at ground potential on feed-hopper side.
6. Feed hopper.

of forms, a common one being the simple roll type. Some information obtained during an investigation of that type of separator, and which indirectly is applicable to other types, is presented here.

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Numerical Relationship

In Fig. 1 material discharged from hopper 6 is fed onto grounded rotating roll 2. Electrode 3 is at a high electrical potential with respect to roll 2, and any conductive particle in contact with roll 2 therefore acquires an electrical charge of such polarity that, when present in the electrical field, the force on it is in the direction of electrode 3. A detailed description of the charging by conductance as utilized in electrostatic separators may be found in a previous publication.[1]

As a result of this force, the path of the projected particle is changed in increasing amounts in the direction toward region *C* as the magnitude of the charge is increased. With the dividing edge 4 the projected particles may be divided into two fractions—those in region *C* designated as "deflected" or "deflectant" and those in region *B* designated as "residue."

When material is passed over the roll only once, a complete separation is seldom obtained, therefore most commercial separators contain several roll-separator units in series, and some practical machines use as many as 12. In this paper a "pass" means the passage of material over a single-roll unit. The procedure is to repass either residue *B* with a low potential on electrode 3 or deflectant *C* with a high potential on 3. Most electrostatic separations follow a flowsheet of roughing and cleaning either the concentrate or the tailing, or both. In a specific instance, a dry, heated mixture of wet cleaned chromite and garnet from an

[1] References are at the end of the paper.

Oregon beach sand was pretreated in an electric corona discharge from a large number of needle points and then passed through the roll-separator field; the more highly conductive chromite was deflected to a greater extent than the less conductive garnet. The residue was repassed repeatedly, giving a successive number of deflectants containing both chromite and garnet. In curve 1, Fig. 2, the quantity of material deflected versus the pass number is plotted as a semilogarithmic graph. After eight passes the residue amounts to 32 per cent of the original material. In curve 1, drawn as a straight line, only the points for the first four passes lie close to it. Curve 2 shows the results when the material (mostly chromite) deflected from these eight passes is recombined and repassed under the same conditions. Here all the points fall close to a straight line and indicate a simple numerical relationship for a one-component mixture of particles.

Probability of Deflection

The explanation for this fixed numerical relationship is that all the particles are identical, within certain experimental limits, and that each is so constituted physically that its probability of passing the dividing edge equals in value the fraction passing when a large number of particles are considered. Accordingly, the same results are obtained every time the experiment is repeated, even when only the residue is repassed.

The ratio of the number of particles in region C to the number in both regions C and B is then equal in value to the probability of deflection of the particles. Let P represent the probability of deflection, D the total amount of material deflectable at the rate of probability P, and n the pass number. The residue and amount deflected can be tabulated in a step-by-step calculation as shown in Table 1. The amount deflected at pass n equals

$$W_n = P(1 - P)^{n-1}D. \qquad [1]$$

The residue after n passes equals:

$$R_n = (1 - P)^n D. \qquad [2]$$

Taking logarithms of both sides of Eq. 1 and collecting the terms in the desired form, gives:

$$\text{Log } W_n = n \log (1 - P) + \log \frac{PD}{(1 - P)}$$

If $\log W_n$ versus n is plotted on Cartesian coordinates, the slope of the line connecting the points is:

$$\frac{\Delta \log W_n}{\Delta n} = \frac{n \log (1 - P) - (n + 1) \log (1 - P)}{n - (n + 1)} = \log (1 - P) \qquad [3]$$

If P is a constant, the slope is constant, and the line through all of the points is a straight line whose slope is $\log (1 - P)$.

The value of P may be determined easily from the values of W_n at the intersection of the plotted line with two consecutive values of n:

$$\frac{W_{n+1}}{W_n} = \frac{P(1 - P)^n D}{P(1 - P)^{n-1}D} = (1 - P) \text{ or } P = 1 - \frac{W_{n+1}}{W_n} \qquad [4]$$

Table 1.—*Calculation of Residue and Amount Deflected*

Pass No.	Passed	Deflected	Residue
1	D	PD	$(1 - P)D$
2	$(1 - P)D$	$P(1 - P)D$	$[(1 - P) - P(1 - P)]D = (1 - P)^2D$
3	$(1 - P)^2D$	$P(1 - P)^2D$	$[(1 - P)^2 - P(1 - P)^2]D = (1 - P)^3D$
.	.	.	.
.	.	.	.
n	$(1 - P)^{n-1}D$	$P(1 - P)^{n-1}D$	$(1 - P)^nD$

Although no check has been made experimentally, similar equations may be derived when the deflected material is repassed and the residues are weighed.

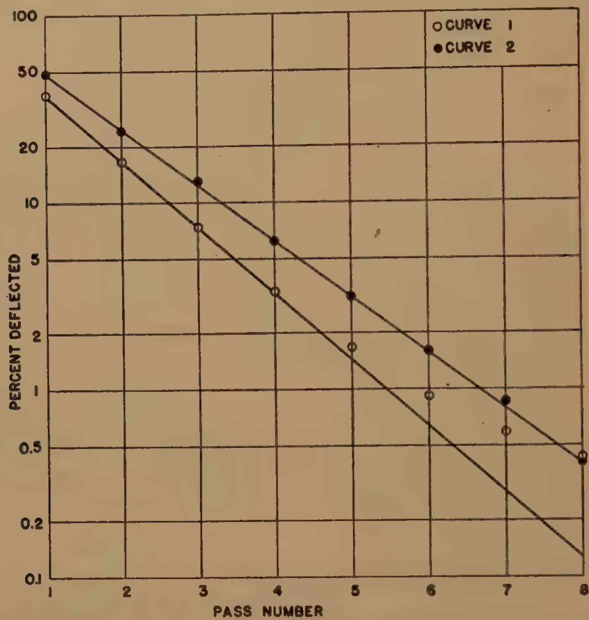

FIG. 2.—RELATION BETWEEN PERCENTAGE DEFLECTED AND PASS NUMBER.

All the points of curve 2, Fig. 2, fall on a straight line because every particle of the material—in this case, chromite—within experimental limits has identical properties. Curve 1 represents a mixture of chromite and garnet. The chromite, which has the highest probability of deflection, determines the percentage deflected in the first passes, since the amount of garnet deflected is negligible. In the latter passes, the chromite in the residue is reduced appreciably, so that the garnet, with the lower probability of deflection, manifests itself. In Table 2 the constant value obtained by taking the differences between the observed values and the straight line of curve 1 represents the amount of garnet in the deflectant.

Similar curves may be constructed theoretically when the probabilities are sufficiently divergent; for example, $P = 0.600$ and 0.002 for two components in a 50–50 mixture. When the probabilities are closer together, the initial probability straight line cannot be drawn, and the deviation in the later passes is not discernible. The presence of a lower-probability component in a mixture of two components whose probabilities are not highly divergent can be detected by the difference in the actual and theoretical residue calculated from the initial probability line by Eq. 2. Information is obtained also by comparing the percentage deflected at the intersection of the straight line at $n = 1$ with the percentage deflected as calculated from its slope.

TABLE 2.—*Garnet Estimation of Figure 2*

Pass No.	Percentage Deflected		
	Actual	Straight Line	Difference or Garnet
5	1.68	1.45	0.23
6	0.92	0.65	0.27
7	0.59	0.29	0.30
8	0.43	0.13	0.30

In actual tests it has been found that the probability of deflection of mixtures, the probability of the components of which are not highly divergent, cannot be calculated as an additive function of the probabilities of the separate components, owing perhaps to interferences in making contact, in movement, or in electric shielding, which are not simply additive. Accordingly, the behavior of only homogeneous one-component powdered materials will be considered in the following discussion.

Charging the garnet by the corona discharge not only decreases the probability of deflection of the garnet but also increases the probability of deflection of the chromite, as illustrated in Fig. 2. The initial probability, representing the probability of chromite, is 0.55 in curve 1 and 0.49 in curve 2. The corona discharge produces a positive charge on the garnet, and the effect is not completely diminished for as long as a 1-hr. interval between the charging and separating operations. The increase in the probability of the chromite due to the presence of the positively charged garnet is logical in view of the warping of the electric field in the vicinity of the garnet, which is charged opposed to deflection.

Probability versus Potential

The question now arises as to the variation of the probability of deflection with respect to the electrical potential between roll 2 and electrode 3 in Fig. 1. Using the same material and the same procedure for obtaining curve 2, Fig. 2, tests were run at different potentials. When the probabilities obtained at these different potentials versus the potential are plotted as a semilogarithmic graph, the potentials being on the logarithmic ordinate, all the points fall close to a straight line. The general equation for such a line is:

$$P = q \log V - r \quad [5]$$

where V is the potential, q and r are constants, and "log" is the logarithm to base 10. The actual and calculated values are listed in Table 3.

Table 3.—*Probability of Deflection versus Potential*

V	P	
	Observed	Calculated[a]
80	0.140	0.140
90	0.294	0.276
100	0.396	0.398
110	0.508	0.508
120	0.597	0.609

[a] From equation of line $P = 2.661 \log V - 4.924$.

In all the tests described the high-voltage supply consisted of a voltage doubler rectifier circuit using two R.C.A. 878 tubes. The input of the high-voltage transformer was varied with a 200-cm. Variac and maintained at a fixed value with a Westinghouse TS voltage regulator. Only the input voltage of the transformer was measured, and a rough calibration of the output voltage with a sphere gap was made. The load was zero except for insulation leakage, and the calibration curve passed closely through the origin. The output voltage was approximately equal to the a. c. input voltage multiplied by 130. The values V given in this paper are the actual measured a. c. input voltages. A more accurate procedure would be to measure the output voltages with an electrostatic voltmeter. The polarity of electrode 3 was positive with respect to roll 2.

With a mechanical rectifier, having a minimum gap distance and accordingly a calibration curve that does not pass through the origin, input voltages cannot be used.

When new units that are proportional to V are used for V, a constant value will be added to r. For logarithms to base e, instead of base 10, q would be divided by 2.30. Values for P less than zero and greater than 1 have no meaning. How close to these limits Eq. 5 is valid is not known.

Table 4 summarizes the data for the values of q and r for various minerals. If the tests for each mineral are extended over a period lasting more than several hours, variations and lack of accurate checks result. The cause of this is not certain. A number of factors may be influential, such as temperature, humidity, chemical and physical changes on the surface of the particles, light, and electrical atmospheric disturbances. The separator and material were heated to prevent any detrimental effects of high humidity.

TABLE 4.—*Constants for* $P = q \log V - r$

No.	Mineral	Source	Setting[a]	q	r
1	Chromite[b]	Rhodesia	*B*	1.17	1.93
2	Ilmenite[c]	India	*B*	1.78	2.92
3	Ilmenite	India	*A*	2.25	4.20
4	Magnetite[c]	California	*B*	2.04	3.29
5	Chromite[d,c]	Oregon	*B*	2.18	3.45
6	Chromite	Oregon	*A*	2.74	5.04
7	Chromite	Oregon	*C*	2.48	4.28
8	Galena[b]	Kansas	*B*	2.49	3.86
9	Chalcopyrite[b]	Quebec	*B*	2.10	2.95

[a] Setting of separator by reference to Fig. 1:

	X	*Y*	*Z*
A	2.1	6.0	3.8
B	2.5	6.4	2.5
C	0.8	6.0	3.8

Rate of feed, approximately 1.8 kg. per min. per meter width of roll.
Roll speed, 24 r.p.m.
[b] Crushed and sized (−32 + 150) Tyler.
[c] Beach sand.
[d] Contains a small amount of ilmenite and magnetite.

Probability and Selectivity

For a particle in contact with the roll there is a resistance to the flow of electrical charge between the particle and roll. Many substances separated by the conductance method are semiconductors.[2] It has been shown that with semiconductors the resistance is asymmetrical, that most of it occurs at the boundary of the semiconductor and electrode,[3] and that it depends on the contact potentials of both the semiconductor and the electrode.[4] For example, with TiO_2 the interfacial resistance of a crystal in contact with a gold electrode is 14 times greater than when it is in contact with a magnesium electrode.[4] With such substances, when the flow of current is in the high-resistance direction, total resistance R is inversely proportional to area S of the high-resistance interface.

For substances that do not come under the class of semiconductors or do not have an asymmetrical high-resistance interface, the resistance is influenced by the small interfacial area. If S_l is the cross-sectional area perpendicular to the direction of flow, l the distance from the roll to some point within the particle, and C the specific conductance, then

$$\int_0^R dR = \frac{1}{C}\int_0^l \frac{dl}{S_l}.$$

The increments of R are small at distances not close to $l = 0$, owing to the larger values of S_l. Most of resistance R therefore occurs at the interface. A fictitious interfacial conductor can now be visualized having a unit length and a cross-sectional area S, which will be called the interfacial area, and

$$S = \frac{1}{CR} \qquad [6]$$

The total conductance is SC. For the high-resistance direction in semiconductors S is the area of the high-resistance interface and C is its conductance per unit area, and for substances that are not semiconductors S is the cross-sectional area of a conductor of unit length, and C is the specific conductance of the material.

Fig. 3 shows three diagrams of two electrodes with a spherical particle adjacent to the lower electrode. The particle can be connected to the lower electrode by means of an electrical conductor, shown here as a line. The electrodes are uncharged in *a* and charged in *b*. These form an electrical condenser in which the electrical charge is proportional to the potential across them.

The spherical particle connected by the bridge is part of this condenser. For convenience, the upper electrode is placed at zero potential and the lower electrode either negative or positive (positive in Fig. 3). Now increase the potential from zero, as in *a*, to some value as shown in *b*. The electrical charge on the particle will change, owing to an electrical current across the bridge. If the potential of the lower electrode is changed by the amount ΔU, the change in charge on the particle will be $K\Delta U$, where K is the electrostatic capacity of the part of the condenser system contributed by the particle. Disconnect the bridge, and again raise the potential of the lower electrode by the amount ΔU. The charge lacking on the particle will be $K\Delta U$, and accordingly the potential difference across the bridge will be ΔU. We have thus shown that the potential across the bridge or between the particle and the lower plate is proportional to the electrical charge on the particle, or, if Q is the total charge on the particle,

$$Q = KU \qquad [7]$$

If in *a* the bridge is first disconnected and the potential raised to a value U, the conditions will be represented by *c*. Here the total charge on the particle is zero, but a partial transferable charge, Q_-, will exist equivalent to KU, with the total charge equal to $Q_- + Q_+$. This transferable charge will flow when the bridge is connected and will change from Q_- to zero, and simultaneously the total charge from zero to Q_+, unless interrupted by the breaking of the bridge. Fig. 3*c* represents the behavior in an electrostatic separator.

With t as the time, the fundamental relation reads:

$$dQ = \frac{U}{R}\,dt$$

U may be eliminated by combining this with Eq. 7, and with the conductance expressed by Eq. 6 we have:

$$\frac{dQ}{Q} = \frac{SC}{K}\,dt \qquad [8]$$

Here the total conductance is independent of the pressure of the particle against the roll. In switch-gear design,[5] it has been

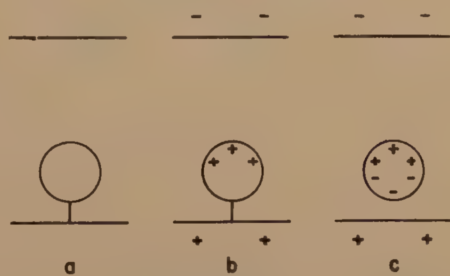

FIG. 3.—PARTICLE CHARGE AND POTENTIAL RELATIONS.

found that the conductance is proportional to the pressure. Eq. 8 would then become

$$\frac{dQ}{Q} = \frac{\epsilon G}{K}\,dt$$

where G is the pressure of the particle against the roll, and coefficient $\epsilon = \frac{1}{RG}$ is probably a function of the material of which the particle is composed.

Integrating both sides of Eq. 8 with a careful selection of limits, the total charge Q on the particle after a charging time t can be expressed by

$$Q = Q_T\left(1 - e^{-\frac{SC}{K}t}\right)$$

where Q_T has the magnitude of the transferable charge. The sign of Q must be determined from the direction of the electric field. At $t = 0$, $Q = 0$, and at $t = \infty$, Q has the magnitude of Q_T. These limits and the rate of change of Q with respect to t can be seen more clearly by plotting on Cartesian coordinates Q versus t as represented by this equation.

The equation just described represents the introduction into the electric field of an initially uncharged particle. If the particle, however, is initially charged, as by a corona discharge, the origin for the t axis

would be shifted by the amount $t_B = \frac{K}{SC} ln\left(1 - \frac{Q_B}{Q_T}\right)$. The shift in time t_B corresponds to changing the initial charge

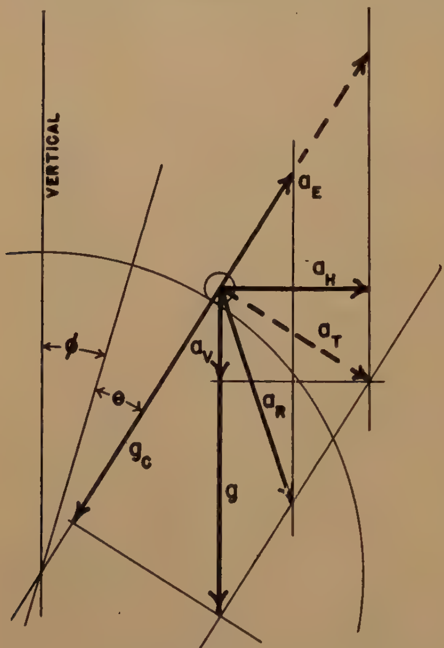

FIG. 4.—ACCELERATIONS ON PARTICLE ON ROLL.

from zero to Q_B. Therefore

$$Q = Q_T\left[1 - e^{-\frac{SC}{K}(t-t_B)}\right]$$

At $t = 0$, $Q = Q_B$ and at $t = \infty$, $Q = Q_T$. The sign of Q_B with respect to Q_T and its limits of variation must be considered. A more simplified form of this equation is

$$Q = Q_T - (Q_T - Q_B)e^{-\frac{SCt}{K}}$$

Fig. 4 represents a particle on the roll of the separator. $(\phi + \theta)$ is the angle between the vertical and the line through the center of the roll and the center of the particle at the position where the particle leaves the roll. ϕ is the angle between the vertical and the position line of the particle where electrical charging becomes large enough for measurement, and may be considered a constant. θ is the angle through which the roll rotates during charging time t_B.

The forces on the particle having mass m are due to the acceleration of gravity g, and the acceleration a_E resulting from charge Q in a field of intensity E. Near the surface of the roll the electric field is perpendicular to the surface, so that a_E is also perpendicular. During the interval t_E, a_E increases from zero to minus g_c. Simultaneously, a_R, the resultant of g and a_E, rotates upward until it corresponds to a_T. After a_E has reached a certain value, the frictional forces will be overcome and the particle will slide on the roll, but it will not leave the roll until a_R corresponds to a_T. When the particle leaves the roll, electrical charging ceases, and a_E reaches a maximum. a_T is always perpendicular to a_E and hence is always tangent to the roll surface. The particles, accordingly, leave the roll tangentially.

This also has been the observed behavior in the tests described. When the rate of charging is very rapid, owing to more intense deflecting fields such as are necessary for strongly adhering corona-charged particles, the trajectory is not tangent and the particles leave with a "trigger-action" release. Probably this is because a current passes even after some very small distance of separation of the particle from the roll, with the result that the competition between the time for the particle to gain momentum and the rate of charging permits the accumulation of a charge in excess of that needed for separation alone.

Particles having different values for the interfacial area S or the coefficient ϵ will have different charging times t_E; accordingly, with a distributed set of values of S there will be a corresponding distributed set of values t_E. The value of a_H at any time t_E or any roll position θ is independent of the electric field intensity, for, as shown by Fig. 4,

$$a_H = g \cos(\phi + \theta) \sin(\phi + \theta)$$

where a_H is the initial horizontal component of acceleration. The value of the horizontal component of acceleration after the particle leaves the roll is complicated by the non-

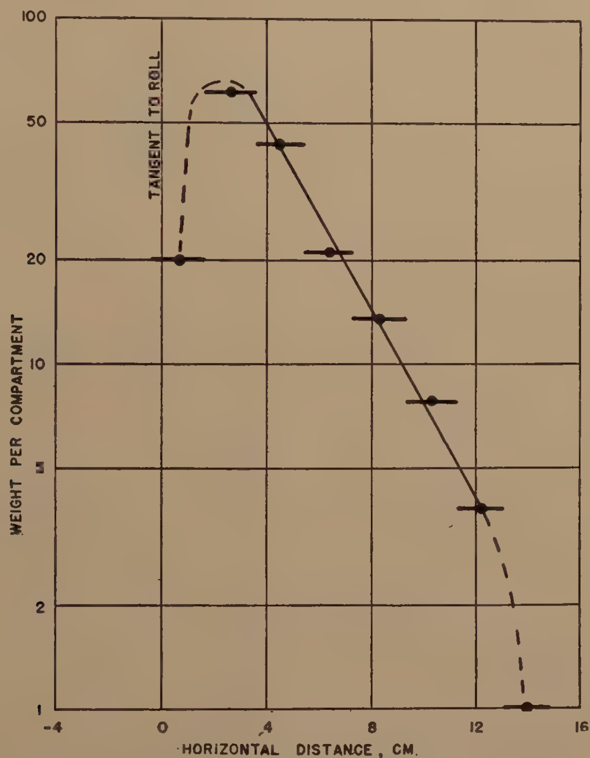

FIG. 5.—QUANTITY DEFLECTED AT VARIOUS DISTANCES.

uniform electric field in conjunction with the path of the particle. For this reason, and because of the fact that the frequency distribution of the values S or ϵ is unknown, it is difficult to calculate the amounts of material projected to various horizontal distances x.

However, it has been found experimentally that the frequency distribution of the horizontal distances is exponential. Fig. 5, a semilogarithmic graph, shows, with electrode setting A and with $V = 115$, the amounts of Oregon beach-sand chromite collected in an eight-compartment box having each compartment at succeeding horizontal distances x from the roll. If W is the weight deposited per unit distance x, the equation of such a line is

$$W = C_A e^{C_B x}$$

where C_A and C_B are constants, of which C_B is negative.

It has been assumed that the particles are homogeneous and that C is constant over the whole particle. If C also varies, SC may be substituted for S, and will vary through a range of values. This is illustrated in the radio crystal detector, consisting of a crystal of galena and a copper-wire "cat whisker." The galena is a semiconductor, and the junction of the wire and the crystal has an asymmetrical conductance. As the wire is moved over the crystal, points of different degrees of "sensitivity" or variations in the value of SC are found.

The value of S and accordingly distance x to which the particle is deflected are probable but not certain. If the particle were in contact not once but many times at

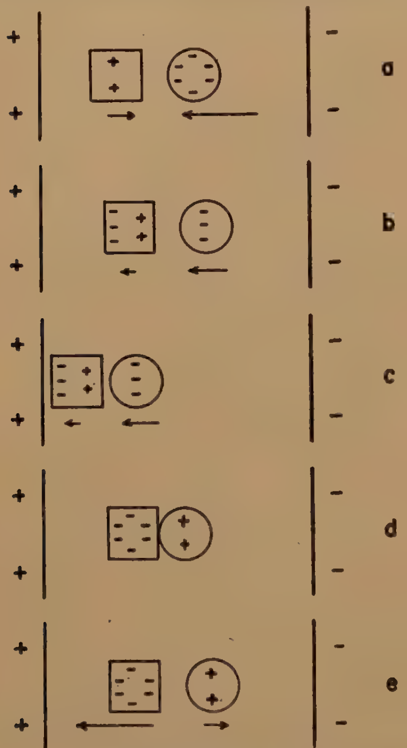

FIG. 6.—REDISTRIBUTION OF CHARGES BY INTERFERENCE.

different points on the particle surface during time interval t_E,[6] an average value of the interfacial area would be obtained. This average area would approach a value that is constant for every particle. When the average area is the same for every particle the path of the particles becomes a certainty and the probability vanishes. Under these conditions the dividing edge can be set so as to give a much sharper separation of particles of different conductances.

Such an improvement is obtained only when the charging and electrostatic deflection occur in separate stages. One procedure is to precharge on a preceeding roll. Thus in a two-roll arrangement the first roll would have electrode 3 in Fig. 1 replaced by needle points at a high potential for an intense corona charging. The particles adhere to this roll and are removed mechanically by a brush, whereupon they descend to the second roll having no needle points but the electrode arrangement of Fig. 1. In the passage from the first to the second roll the particles are brought into contact with the grounded chute and vibrating feeder at a large number of points on their surfaces. Discharging progresses at each point of contact, and with a large number of points of contact the surfaces are averaged.

An alternative is simultaneous charging by ionic emission and discharging by conduction with a suitable ionic emitter placed above feeder 1.

An illustration of a separator in which the behavior of a particle is almost a certainty is the magnetic separator. In this separator the magnetic force is determined from the total volume of the particle and not from a probable interfacial area. Hence, the separation that can be effected in the magnetic separator is usually accomplished in two passes.

However, averaging the interfacial areas will not completely eliminate the sources of poor selectivity in the conductance separator, because of the inherent characteristic of the conductance of the particle itself. In heavily loaded separators where one component of a mixture must pass through a cloud of suspended particles of the other component or through a heavy layer of the other component while still on the roll the particles of the two components will frequently be in contact with each other. Since the particles are both conductive, redistribution of electrical charges on contact and complete nullification of the intended separation will result, as illustrated in Fig. 6. At a two particles are traveling in their proper opposite directions, at b a fast particle (round) is attempting to pass a slow particle, and at c two particles charged by a

corona discharge are deposited on the roll, one on top of the other. A collision results, and the charges are redistributed as at d. At e, according to this redistribution, the particles travel in directions opposite to the direction that would yield good separation. The only way in which this interference can be eliminated is to narrow the thickness of the cloud in the direction of separation and have the two components travel in opposite directions, or to thin out the cloud or layer on the roll with a decreased load and increased velocity of passage through the separator.

Summary

When homogeneous particles are separated on a simple roll-type separator and the successive residues repassed, the weights of material deflected will fall on a straight line if plotted versus the pass number as a semilogarithmic graph. The amount of material deflected at pass n is:

$$W_n = P(1 - P)^{n-1}D$$

when n, P, and D are, respectively, the pass number, the probability of deflection, and the total amount of material deflected at the rate of probability P.

The probability varies with the electrical potential according to the equation

$$P = q \log V - r$$

where V is the potential and q and r are constants that depend on the units of V, the dimensions of the apparatus, and the conductance of the particles.

The amounts of material W deflected at various horizontal distances x from the roll can be represented by the equation

$$W = C_A e^{C_B x}$$

where C_A and C_B are constants, of which C_B is negative.

After some consideration of what actually happens during electrostatic separation, the conclusion is reached that the interfacial area or the interfacial conductance has a different value every time the particle is in contact on the roll. Behavior of particles in the conductance electrostatic separator is therefore a probability and not a certainty.

A method is suggested for improving the selectivity of the conductance separator.

Acknowledgment

The author is greatly indebted to Mr. O. C. Ralston for his encouragement and contributions and to Dr. P. S. Roller for his criticism.

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DISCUSSION

(Charles E. Locke presiding)

H. L. Bullock, New York, N. Y.—Mr. Fraas' paper is a valuable addition to the scant store of exact data we have on the subject of electrostatic separation. Its value is twofold in that it gives, first, mathematical formulas that may be used to predict results and to check the efficiency of commercial separations and, second, it serves to focus our attention on the conditions that may be modified to increase the probability of deflection and to provide more selective separations.

On page 579 Mr. Fraas makes one statement with which I cannot agree. In the second paragraph he says: "Charging the garnet by the corona discharge not only decreases the probability of deflection of the garnet but also increases the probability of deflection of the chromite, as illustrated in Fig. 2. The initial probability, representing the probability of chromite, is 0.55 in curve 1 and 0.49 in curve 2. The corona discharge produces a positive charge on the garnet, and the effect is not com-

pletely diminished for as long as a 1-hr. interval between the charging and separating operations. The increase in the probability of the chromite due to the presence of the positively charged garnet is logical in view of the warping of the electric field in the vicinity of the garnet, which is charged opposed to deflection."

While it is true that the electrostatic field will be warped in the vicinity of the positively charged garnet, and more lines of force will be crowded on to the chromite, it is also true that there will be an electrostatic field built up between the positively charged pieces of garnet and their neighboring pieces of negatively charged chromite. The potential difference, of course, will be much lower than between the chromite and the electrode but the distance over which this secondary field acts is so much smaller that there is actually a restraining effect produced by the garnet. This seems to be borne out by Fig. 2, in which curve 2 is shown at a higher level than curve 1.

According to the definition given on page 577, the probability of deflection is the ratio of the number of particles in region *C* to the number of particles in both regions *C* and *B*. In other words, the probability of deflection is equal to the ratio of the deflected material to the total of deflected material plus residue material. In the first pass, the probability of deflection, therefore, is equal to the percentage deflected and, from Fig. 2, the percentage of material deflected on the first pass from the mixture of chromite and garnet is approximately 42 per cent, while the percentage of material deflected from the practically pure chromite concentrate appears to be just under 50 per cent. Consequently, the probability of deflection increased from 42 to 49 or 50 per cent upon removal of the garnet, although other conditions of operation were kept exactly the same.

The commercial operator is interested in the maximum recovery of the purest possible product and will, therefore, wish to have all factors arranged for the most favorable action. Mr. Fraas has mentioned pretreatment by corona discharge, reduction in thickness of the layer of material going through the machine and increased velocity of passage through the separator. Reduction of thickness of the layer cuts down on the capacity of the machine and the increase in velocity is employed in an effort to compensate for this. Actually, there is a definite limit to the possible increase in velocity, as time is required for the neutralization or draining off of the charge on the conducting particles. It should be noted at this point that there is an automatic thinning out of the material in commercial multipass separators, due to the removal of the deflected material from the feed and the passage of the residue over successive separating stages of identical physical dimensions. The reduction of feed density is very evident in cascade machines of the three-stage type, where the top section is operated at high potential or reduced clearance to produce a rough split of deflected and residue material, and these two products are then passed through two lower sections of equal size to produce a clean concentrate, a tailing suitable for rejection and a combined central middling product suitable for further treatment.

As Mr. Fraas' paper is concerned primarily with the mathematical probability of electrostatic separation, naturally he could not go into great detail in developing the factors that tend to make the behavior of the particles a probability rather than a certainty. However, it is evident that he has recognized them fully as, in the paper on Electrostatic Separations of Solids prepared by Mr. Fraas and Mr. Oliver C. Ralston,[1] under the heading Conductance Separator, we find the following statement: "The magnitude of the electrostatic movements due to easy production of large charges screens out observation of other less prominent phenomena."

I believe that the commercial operator of electrostatic separators is resigned to the fact that the removal of a deflectant is subject to the laws of probability as developed by masking of the deflectant particle by a preponderance of residue in successive passes. He is willing to use more passes and to leave some value in the tails, if the concentrate comes clean. It is the probability of residue or tails coming over in the deflectant or concentrate that causes trouble.

Where a separator is set up with the proper voltage, correct spacing of dividers and attracting electrodes, and the right velocity of feed, it is probable that any appreciable contamination of the concentrate is due to these "other less prominent phenomena." These "other less prominent phenomena" may be arranged to work with or against the conduc-

tion action. They are present because it is impossible to construct a separator that will operate on a mixture of two or more materials by the principle of conductance alone. In all conductance separators, the movement of the materials on each other and on the rolls or chutes of the separator is constantly generating surface contact charges. If these charges reinforce the charge on the nonconducting residue and neutralize the charge on the conducting deflectant, they aid the degree and purity of separation. If they neutralize the charge on the nonconducting residue and reinforce the charge on the conducting deflectant, they lead to reduced efficiency and an impure concentrate. This point was developed in my paper, Scope and Economics of Electrostatic Separation,[7] in which, under the heading of The Apparatus, I said, "For the ideal condition the contact potential of the supporting surfaces should lie between the contact potentials of the ingredients of the mix, so that the resulting surface potentials of the ingredients will be opposite and at a maximum. This condition is also advantageous where the separation depends primarily on conduction charges, for then the conducting particle will have the same sign as the feed roll or chute, and the production of a maximum charge of opposite sign on the nonconducting constituent will make it stick to the roll or chute and improve the efficiency of the separation. Since it is possible to reverse the polarity of the charge on the electrode and frame of the separator, it is always possible to operate the separator with the frame bearing a charge opposite to the contact charge carried by the nonconductor."

Consideration of what happens in successive passes in a conductance separator makes the proper selection of polarity even more important. As the material passes through successive stages, the conducting particles quickly lose their charges to the rolls and chutes, at the same time the surface charges of the nonconducting particles are being slowly reduced by repeated contact with the metal surfaces, and as the conducting particles take on the polarity of the rolls and chutes, they furnish additional contacts to drain the charges from the nonconductors. This drainage becomes serious with numerous passes unless it is counteracted by the production of the proper contact charges. As the contact charges are produced continuously by the movement of the material through the separator, their selective use can be employed to greatly improve the separation.

As the conductance separator is subject to the gradual reduction of the pretreatment effect of the corona discharge, and as the separation is dependent on the time interval for the loss of charge by the conducting particle, it is to be hoped that Mr. Fraas will have the opportunity to develop probability relations for the contact-charge type of separator so that we will have a mathematical basis for comparison of the two types of separator.

F. Fraas (author's reply).—Mr. Bullock is in error in his deduction of the probabilities represented by Fig. 2. The probability of deflection for the chromite must be determined from the slope of the straight line and not from the intercept at the first pass. A greater slope for this line corresponds to a higher probability of deflection of the chromite. The slope of the straight line is larger in curve 1 than in curve 2, and the effect of the presence of the garnet, which is charged to a polarity opposed to deflection, is, therefore, to increase the probability of deflection of the chromite.

[7] H. L. Bullock: Scope and Economics of Electrostatic Separation. *Ind. and Eng. Chem.* Ind. Ed. (1941) **33**, 1119.

Climax Milling Practice

By E. J. Duggan,* Member A.I.M.E

(Salt Lake City Meeting, September 1940, and New York Meeting, February 1942)

The mine and mill of the Climax Molybdenum Co. are at Climax, Colorado. Climax is on Fremont pass directly on the Continental Divide, at an elevation of 11,400 feet.

Description of Ore

The Climax ore-bearing rock is essentially an altered and highly silicified granite, fully half of the gangue being quartz. Molybdenite is the only mineral of economic consequence and most of it is intimately associated with quartz in the form of fine veinlets and stringers. Other minerals are molybdite, pyrite, and chalcopyrite. While the molybdite may be readily extracted by hydrometallurgical methods, its content is too low to make such operations profitable. The pyrite content ranges from 2 to 5 per cent and that of the chalcopyrite from 0.03 to 0.05 per cent. There is not enough copper to make its recovery attractive and one of the metallurgical problems is the elimination of it and the pyrite from the product.

On account of the close association of the molybdenite with the quartz and the necessity of eliminating the pyrite and chalcopyrite from the final product, fine grinding followed by flotation is the only feasible method of concentration.

History

Construction of the first milling unit was started in 1917 and it was operated during parts of 1918 and 1919.

In the postwar years, there was almost no demand for the product and production was at a standstill until 1924. The Company's research campaign to develop the use of the metal began to show results in that year. From 1926 to 1932 production ranged from 500 to 1200 tons per day. In 1931 an additional mill section was built and production was gradually increased until in 1934 and 1935 about 3000 to 4000 tons was milled per day. This tonnage overtaxed the milling facilities, and as the demand for molybdenum was growing constantly, additional milling equipment was needed. In 1936 and 1937 another crushing plant, six additional mill sections and the necessary auxiliary plants were constructed and brought into operation.

This plant was designed to mill 10,000 tons per day, but it was not long before 12,500 tons was being treated during periods of peak demands. During the summer of 1941, to meet defense needs, the mill has been required to treat 15,500 tons per day. This has been accomplished with little plant expansion and at no loss in recovery, as will be described later. To increase capacity still further, one more primary section is now being built. This will bring capacity to 18,000 tons per day, which can be stretched to 20,000 tons at a small sacrifice in recovery.

Crushing

There are two crushing plants: No. 1, of 5000 tons capacity (flowsheet in Fig. 1) and No. 2, of 15,000 tons capacity (Fig. 2).

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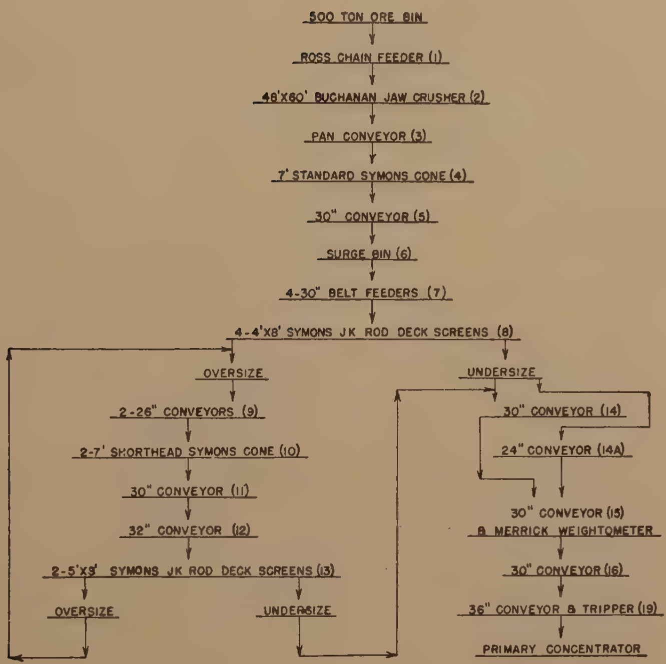


(1) ROSS FEEDER -- 15 HP MOTOR
(2) 48" X 60" BUCHANAN JAW CRUSHER -- 130 RPM, 300 HP MOTOR -- SETTING 9"
(4) 7' SYMONS STANDARD CRUSHER -- 235 RPM, 300 HP MOTOR -- SETTING $1\frac{1}{2}$"
(6) SURGE BIN
(8) 4 - 4'X8' SYMONS JK ROD DECK SCREENS -- $\frac{5}{16}$" X 7" OPENING, $\frac{3}{16}$" ROD, 1325 STROKES / MIN., $\frac{3}{8}$" STROKE, $7\frac{1}{2}$ HP MOTOR
(10) NO. 2 & 3 -- 7' SYMONS SHORT HEAD CRUSHER -- 230 RPM, 300 HP MOTOR, SETTING $\frac{1}{4}$"
(13) 2 -- 5'X8' SYMONS JK ROD DECK SCREENS -- $\frac{5}{16}$" X 7" OPENING, $\frac{3}{16}$" ROD, 1325 STROKES / MIN., $\frac{3}{8}$" STROKE, $7\frac{1}{2}$ HP MOTOR

CONVEYORS.
(3) PAN CONVEYOR -- LENGTH 58' (18' HORIZONTAL, 40' ON SLOPE), RISE 19'-0", 60 FPM, 40 HP MOTOR
(5) NO. 2 - WIDTH 30", LENGTH 126', RISE 40', 350 FPM, 40 HP MOTOR
(7) 4-30" BELT FEEDERS 3'-8" LONG, O-RISE. 2&4 29.2 FPM. 1&3 31.8 FPM. 5 HP MOTOR.
(9) NO. 3 - WIDTH 26", LENGTH 100'-6", RISE 20', 395 FPM., 15 HP MOTOR
(9) NO. 4 - WIDTH 26", LENGTH 85'-0", RISE 25', 375 FPM., 15 HP MOTOR
(11) NO. 5 - WIDTH 30", LENGTH 37'-6", RISE 0, 505 FPM., 10 HP MOTOR
(12) NO. 6 - WIDTH 32", LENGTH 61'-6", RISE 21', 425 FPM., 25 HP MOTOR
(14) NO. 7 - WIDTH 30", LENGTH 35', RISE 0', 325 FPM., $7\frac{1}{2}$ HP MOTOR
(14A) NO. 7A - WIDTH 24", LENGTH 22'-10$\frac{3}{4}$", RISE 0', 206 FPM., 5 HP MOTOR
(15) NO. 8 - WIDTH 30", LENGTH 90'-6", RISE 31', 445 FPM., 25 HP MOTOR.
(16) NO. 9 - WIDTH 30", LENGTH 98'-0", RISE 34', 325 FPM., 25 HP MOTOR

VENTILATING FANS

1. NOR-BLO STRAIGHT BLADE EXHAUST FAN SIZE 37-31W-6 OD. TYPE DC-28-HS. SERIAL 935-26. 30200 CU. FT. PER MINUTE. 7" STATIC PRESSURE. 60 HP MOTOR. - 1170 RPM. FAN 810 RPM. (NORTHERN BLOWER COMPANY)

FIG. 1.—FLOWSHEET OF NO. 1 CRUSHER.

The No. 1 plant is used only during periods of peak capacity.

The mine ore, approximately of a size to pass a 36-in. grizzly, is broken to about 9 in. in 48 by 60-in. jaw crushers. This is reduced in size in 7-ft. standard Symons cones to 1½ in. and finally to ⅜ in. in 7-ft. short-head Symons cones. The latter are in closed circuit with vibrating screens. A screen analysis of the crusher product is given in Table 1 and various operating data in Table 2.

Various types of vibrating screens have been tried and all have been successful on dry ore. During the summer months the ore is wet and sticky and on this ore the screens with rod decks have been the only ones wholly free from blinding.

A magnetic iron detector, designed by H. E. Wurzbach, of the Utah Copper Co., is used on the conveyor taking the jaw-crusher discharge. In operation this shuts down the conveyor when a piece of magnetic steel or iron passes under it. It is invaluable in keeping tramp iron out of the cone crushers. The conveyors ahead of all cones are equipped with magnetic head pulleys, but these are not nearly so effective as the iron detector. Overhead magnets have proved to be of little value.

The ore from each plant is weighed on a separate Merrick weightometer.

Dust control has been given great consideration and elaborate equipment has been provided. This consists essentially of a ventilation system by means of which the dust is gathered at the point of generation and discharged from the buildings. Wet and dry cyclones and canvas filters are employed for cleaning the exhausted air, but in no case is the cleaned air returned to the buildings, on account of possible contamination. To replace the air exhausted in this manner and to maintain the inside pressure equal to that outside, wall fans are used to blow clean air into the buildings. For successful dust control all machines, conveyor transfer points and other places that produce dust must be sufficiently enclosed so that a slight negative pressure is maintained within the enclosure by the exhaust system.

About 100,000 cu. ft. of air per min. is exhausted from the main crushing-plant buildings and from the conveyor galleries. Fresh-air fans feed 58,000 cu. ft. per min. to No. 2 plant and similar fans of suitable capacity are planned for No. 1 plant.

Water sprays are used at all transfer points. A vacuum cleaning plant is being installed to clean floors, walls and steelwork and every precaution is taken to secure safe working conditions.

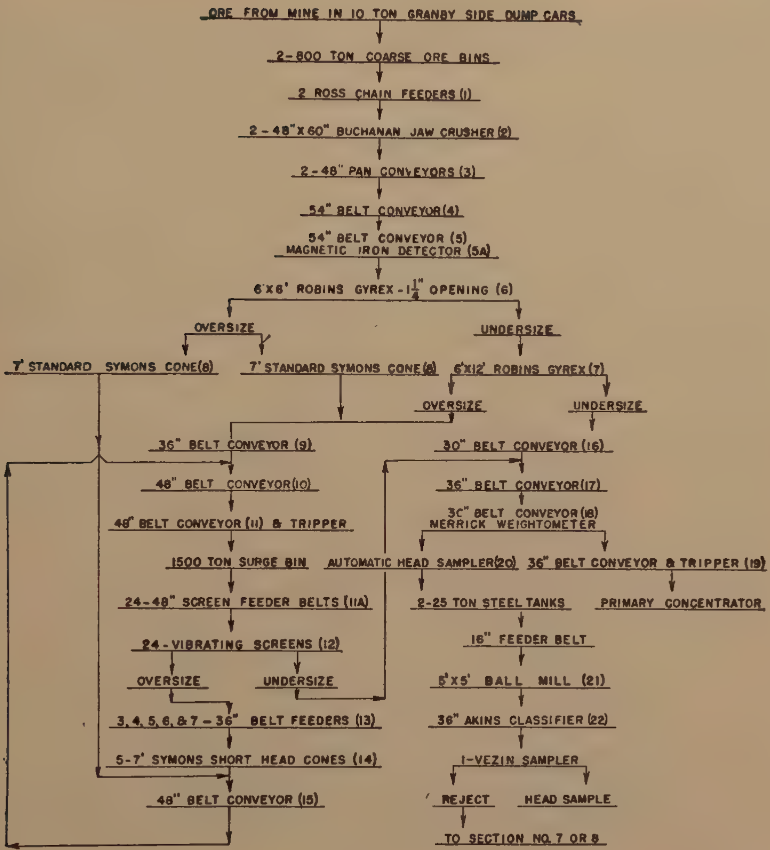
Periodical inspection of all dust-control equipment and dust-count surveys are made by an engineer under the supervision of the Safety Department, to see that all equipment is in good running order at all times.

Fine Grinding and Classification

There are eight primary fine-grinding sections, each independent of all the others. One of these, No. 1 unit (flowsheet, Fig. 3) is the original mill and is used only when the plant is on full capacity. The other seven sections (Fig. 4) are identical, and only they will be described.

The ball mills in each were originally 9 by 9-ft. Marcy mills of the overflow type, directly connected by herringbone gears to 450-hp. synchronous motors. These mills have been converted recently to 9 by 8-ft. grate-type discharge. Each mill is in closed circuit with a 78-in. Akins duplex high-weir classifier.

For the first year or two of operation, 1936–1937, after the installation of the last six sections, each fine-grinding unit produced about 1300 tons per day through 28 mesh. Principally as a result of a careful study of ball sizes, this was brought up to 1550 tons per day. Early in the summer of 1941, to meet the demand caused by the present emergency, the tonnage per section was increased to 1850 tons per day, or



(1) 2 ROSS FEEDERS--15 HP MOTOR, 6-3" CHAIN
(2) 2 BUCHANAN JAW CRUSHER 130 RPM, 300 HP MOTOR -- SETTING 9"
(3) PAN CONVEYORS-- LENGTH 13'6", 26 FPM, RISE 0', 10 HP MOTOR
(6) ROBINS GYREX NO.1-1 1/4" GRIZZLY 10 HP MOTOR
(7) ROBINS GYREX NO.2 1/2 X 3" OPENING 10 HP MOTOR
(8) 2 STANDARD SYMONS CONE CRUSHERS. 235 RPM. 300HP MOTOR. SETTING 1 1/2"
(12) VIBRATING SCREENS:
- 9 ROBINS GYREX SCREENS-5'X10', 5/16"X4" OPENING, 19° SLOPE, 1000 STROKES PER MINUTE, 1/4" STROKE, 7 1/2 HP MOTOR.
- 2 JK SYMONS-4'X8', 5/16" OPENING, 6° SLOPE, 1325 STROKES PER MINUTE, 3/8" STROKE, 7 1/2 HP MOTOR
- 12 JEFFREY TRAYLORS-4'X7', 5/16" X1" OPENING, 25° SLOPE, 1/16" STROKE
- 1 ALLIS CHALMERS-4'X8', LOW HEAD, 5/16"X 4 1/4" OPENING, 1100 STROKES PER MINUTE, 1/2" STROKE, 0° SLOPE, 5 HP MOTOR

(14) 5 SHORT HEAD CRUSHERS, 230 RPM, 300 HP MOTOR, SETTING 1/4"
(20) 1 HEAD SAMPLER -- CUTS 1.0% OF MILL FEED -- 135 FPM, 5 HP MOTOR
(21) BALL MILL -- 28 RPM, 60 HP MOTOR
(22) AKINS DUPLEX CLASSIFIER -- 36" SPIRAL, 5 RPM, 3 HP MOTOR

FIG. 2.—FLOWSHEET OF NO. 2 CRUSHER.

15,000 to 16,000 for the entire plant. This has been on the same size of feed, ground to the same size.

The principal change made to achieve the last increase in capacity was the conversion of the 9 by 9-ft. overflow mills to 9 by 8-ft. grate mills. This was done easily and inexpensively by casting pulp elevators and grate supports and bolting them to the inside of the old discharge heads. This caused a reduction in inside mill length of 10 in., but even so the mill capacity was increased 18 per cent. Power and steel consumption per ton remained about the same, so there has been little or no increase in mill efficiency. Fortunately, enough excess capacity had been built into the motors to handle the increased power load.

In the first test with grates the slots had $\frac{3}{8}$-in. openings but with these openings, it was impossible to get sufficient flow of pulp through the mills to keep the circulating load over 200 per cent and there was no increase in mill capacity from their use. Grates with wider slots, $\frac{3}{4}$-in. openings, are now used. These permit the passage of sufficient pulp to maintain the circulating load at 400 per cent and have increased the capacity 18 per cent.

The mills are lined with Marcy-type shiplap liners of chrome-molybdenum cast steel. The less expensive rail liners are not

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FIG. 2.—(*Continued.*)

BELT CONVEYORS:

(4) NO. 21A -- LENGTH 12'10" RISE 0', 125 FPM, 15 HP MOTOR, 54" BELT
(5) NO. 21 -- LENGTH 204' RISE 52' 0" 220 FPM, 100 HP MOTOR, 54" BELT
(9) NO. 22 -- LENGTH 165' RISE 14' 2" 310 FPM, 30 HP MOTOR, 36" BELT
(10) NO. 24 -- LENGTH 59' 0" RISE 17' 4" 400 FPM, 60 HP MOTOR, 48" BELT
(11) NO. 25 -- LENGTH 311' RISE 50' 6" 430 FPM, 200 HP MOTOR, 48" BELT
(11A) 24 SCREEN FEEDERS -- WIDTH 4', LENGTH 4', RISE 0, 13 FPM, 1 HP MOTOR 48" BELT
(13) NO. 25 -- 3, 4, 5, 6, & 7 -- LENGTH 32', RISE 0, 210 FPM, 3 HP MOTOR 36" BELT
(15) NO. 23 LENGTH 298' RISE 28' 375 FPM 100 HP MOTOR 48" BELT
(16) NO. 26 -- LENGTH 42' 8" RISE 9' 0" 300 FPM 5 HP MOTOR 30 BELT
(17) NO. 27 -- LENGTH 290' RISE 33' 300 FPM 60 H P MOTOR 36" BELT
(18) NO. 28 -- LENGTH 364' RISE 13' 8" 300 FPM 40 H P MOTOR 36" BELT
(19) NO. 10 -- LENGTH 352' RISE 9' 0" 380 FPM 50 HP MOTOR 36" BELT

VENTILATING FANS NO. 2 CRUSHER
MAIN BUILDING

STURTEVANT SILENT VANE - SIZE 105 - CLASS NO. 4 - DESIGN NO. 7
SERIAL 340487 - 40000 CU FT. PER MIN. $6\frac{1}{2}$" STATIC PRESSURE AT 11500' ALTITUDE. MOTOR 50 HP. 1165 RPM FAN RATING 1110 RPM. ACTUAL 1088 RPM.
CLARAGE FAN - SIZE $3\frac{7}{8}$" - TYPE W - DESIGN 3 - SERIAL 62422
34000 CU. FT. / MIN. - $1\frac{1}{2}$" STATIC PRESSURE - MOTOR 15 HP 1180 RPM - FAN 662 RPM
STURTEVANT PLANOVANE EXHAUSTER - SIZE 60 - DESIGN 3 - SERIAL 340488
12600 CU. FT. / MIN - $2\frac{1}{2}$" STATIC PRESSURE - MOTOR 15 HP - 1202 RPM FAN 789 RPM

CONVEYOR GALLERIES

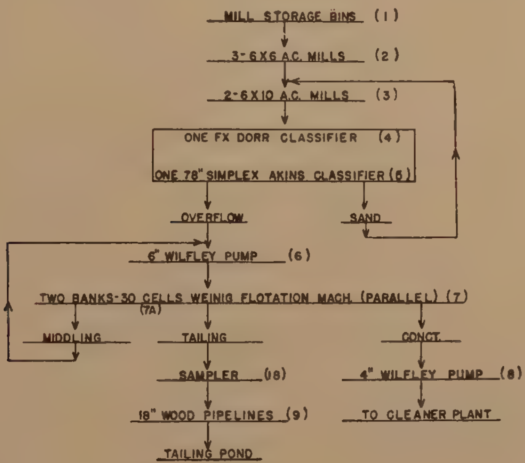
STURTEVANT REX VANE RB1 - DESIGN 3 - SERIAL 324507 - 2650 CU. FT. / MIN. 2" STATIC PRESSURE - MOTOR 2 HP - 1777 RPM - FAN 2462 RPM
2 - STURTEVANT REX VANE RB 2 - DESIGN 3 - SERIAL 324506 — 2970 CU. FT PER MIN. — 2" STATIC PRESSURE — MOTOR 2 HP 1767 RPM — FAN 2180 R.P.M

WALL FANS

STURTEVANT 42" PROPELLOR FANS - DESIGN 7 - 14500 CU. FT. PER MIN MOTOR $\frac{3}{4}$ HP — DIRECT CONNECTED TO FAN — 480 RPM.

used because of loss in mill diameter and time required for relining. The mills must be lined in place and a set of shell liners is replaced in 6 hr. Since capacity is para-

operated at 20 r.p.m., at which speed the ball-mill motors are loaded to capacity, now that the mills are equipped with grates.



NO. I UNIT
SECTION I

(1) RECTANGULAR WOOD BIN 15'X 53'X 28'.
(2) THREE 6X6 ALLIS CHALMERS BALL MILLS - 25RPM - 125 HP. OVERFLOW.
(3) TWO 6X10 ALLIS CHALMERS BALL MILLS - 25RPM - 150HP. OVERFLOW.
(4) ONE FX DORR CLASSIFIER (NO.5 6X10) - 17.3 STROKES PER MINUTE - 8'X 27' - 15HP.
(5) ONE 78" AKINS SIMPLEX CLASSIFIER (NO. 4 6X10) - 4.3 RPM - 15HP - 3.5"/FT. SLOPE.
(6) TWO 6" WILFLEY PUMPS - DIRECT CONNECTED - 875 RPM - 30 HP.
(7) FOUR 15 CELL 36" WEINIG FLOTATION MACHINES - 21" RUBBER IMPELLOR - 300RPM - 5 HP.
(7A) ONE 14X36 ROOTS CONNERSVILLE BLOWER - 338RPM - 50HP - 2.2 LBS PRESSURE.
(8) TWO 4" WILFLEY PUMPS - DIRECT CONNECTED - 1175 RPM - 30 HP.
(9) TWO 18" DIAMETER WOOD PIPELINES - EAST 15,200 FT. - WEST 16,700FT

FIG. 3.—FLOWSHEET OF NO. 1 UNIT, PRIMARY CONCENTRATOR.

mount, the lightest shell liners practical are used. The liners now in use make the average diameter of a mill about 2 in. more than that of a mill as originally lined. This alone makes the tonnage 5 per cent more than formerly.

The mills have been driven at various speeds from 17 r.p.m. to 22 r.p.m. Allowing for wear of shell liner, this has given speeds from 66 to 84 per cent of critical. The higher speeds have yielded greater tonnages, but not in proportion. As the power increases directly with the speed, the lower speeds are more efficient per unit of power. Liner consumption is somewhat less at lower speeds. Ball consumption was found to be the same at all speeds. The mills are now

Various ball sizes and combinations of sizes have been tried, from 2 in. to 4½ in. The standard size is now 3 in. Forged-steel balls are used, as cast-iron balls available at Climax cannot compete in cost. The standard ball is alloyed with 0.2 to 0.30 per cent Mo, which has improved the wearing quality from 15 to 20 per cent.

The ball mills are fed by means of double-compartment scoops, of 5-ft. radius and 18 in. wide.

The classifiers are set at a slope of 3½ in. per foot, which allows the circuit to be closed without any auxiliary lifting device. The spirals turn 2½ r.p.m., which furnishes an average circulating load of about 400 per cent when the machines are fully

loaded. The classifiers have operated at speeds as high as 6 r.p.m. with circulating loads up to 600 and 800 per cent, but the small increase in tonnage did not compen-

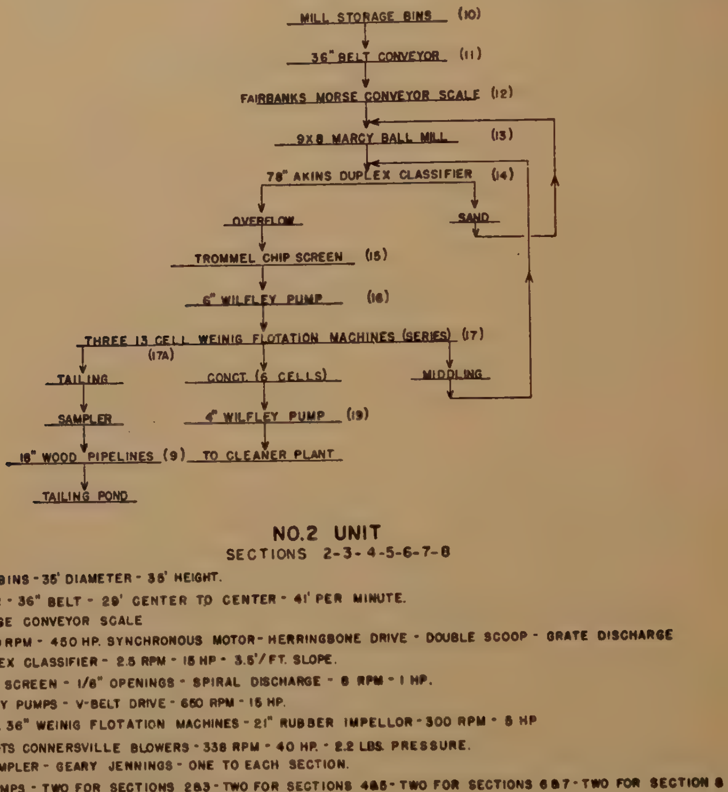


FIG. 4.—FLOWSHEET OF NO. 2 UNIT, PRIMARY CONCENTRATOR.

sate for the increased cost of classifier operation. On the present tonnage of 1850 per day per section, the classifier overflow has a solid content of 45 per cent and is ground to pass 28 mesh, with 40 per cent on 100 mesh.

The feed to each 9 by 8-ft. ball mill is weighed on a Fairbanks Morse conveyor scale, which is an invaluable adjunct for test work.

Screen analyses of ball-mill feed and classifier products are shown in Table 1 and in Fig. 5 is charted the effect on capacity of varying the fineness of the grind.

FLOTATION

Flotation equipment in each primary section consists of 30 Weinig subaeration cells in series. As originally installed there were three machines of 10 cells each in parallel, each machine treating one third of the feed from a section. Putting all the cells in séries increased recovery about 1 per cent at no increase in cost. The probable reason for this is that the cells are of the "hog-trough" type and the all-cells-in-series arrangement overcomes the effects of by-passing. Nine additional cells per section are now being added, to keep pace with the increased output of the ball mills. Air at 1¾ lb. per sq. in. is used at the rate of 35 cu. ft. per min. per spindle.

The froth from the first 6 to 10 cells of a section is removed as a rougher concentrate and the froth from the remaining cells is returned to the feed as a middling.

Reagents are now employed in the primary rougher circuit as follows: pine oil, 0.20 lb. per ton; saturated hydrocarbon, 0.60 lb. per ton.

No cyanide or lime is added directly to the rougher machines, but enough is sent to the primary concentrator in the reclaimed water from the cleaner plant to drop about 75 per cent of the pyrite and copper in the rougher feed.

Two factors have a predominating influence on Climax metallurgy, each tending to counterbalance the other in the present method of attack:

1. A grind of through 270 mesh must be made before satisfactory separation of the mineral from the gangue is attained. In Table 3 is shown the fineness of grind required to effect varying degrees of separation.

2. Molybdenite is one of the most readily floated of the minerals. Under certain conditions middling grains 65 to 100 mesh in size, containing less than 1 per cent molybdenite, can be floated.

In early practice the ore was ground through 100 mesh for rougher flotation and pine oil was usually the only promoter used. Recovery was satisfactory but grade of concentrate was difficult to maintain, as regrinding of concentrate was not employed. Because of the hardness of the ore, cost of grinding was expensive and the capacity of the fine-grinding mills was low.

Later practice has been to make a rough concentrate containing more included mineral at a relatively coarse grind and to confine the expensive fine grinding to a small part of the tonnage. Reagents have been developed that more effectively promote the flotation of the middling grains. This has permitted the milling of relatively high tonnages in the primary fine-grinding circuit, while it has increased the amount of work required to regrind and clean the rougher concentrate. Finer grinding in the rougher circuit than now practiced would lead to a higher recovery, but at a cost in capacity.

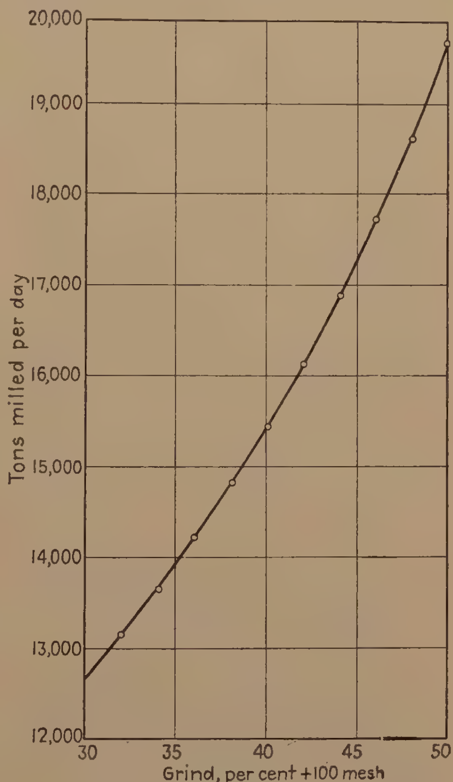


FIG. 5.—EFFECT ON CAPACITY OF VARYING FINENESS OF GRIND.
Total all sections 1 to 8. Tons per 24 hours.

Early test work disclosed that petroleum products had a strong promoting effect on molybdenite, especially on the coarse middling grains. The first test work was confined to the various crudes and their cheaper distillation products. These yielded increased recoveries, but difficulty was experienced in maintaining grade and purity of concentrate, especially through contamination with pyrite and chalcopyrite. The white medicinal oils sold usually under the name of Russian mineral oil were found to be superior to anything else in

the flotation of coarse middling and in the production of a satisfactory concentrate. As these oils are prohibitive in price, a search was made for a cheaper product approaching them in physical properties, and a petroleum fraction was found that yields results closely approaching those obtained with the more expensive mineral oils. The use of this hydrocarbon has increased recovery by 5 to 7 per cent, the effect being more beneficial on coarser grinds.

All the commonly used chemical reagents have been tested, but none have yielded results as good as those now obtained.

A screen analysis of the rougher tailing is given in Table 1.

Cleaner Plant

The following is a typical analysis of the current rougher concentrate that is the feed to the cleaner plant:

	Per Cent
Free molybdenite	5.0
Free pyrite	4.0
Free chalcopyrite	0.5
Free gangue	15.5
Included molybdenite	7.0
Included gangue	68.0

The rougher concentrate averages 12 per cent molybdenite, which means that about 60 per cent of the molybdenite floated in the primary circuit is a true middling.

No attempt is made to float the free molybdenite from the rougher froth before regrinding. This is impractical because of the great ease with which molybdenite included in sands around 100 mesh will float along with the fine free molybdenite.

The cleaning operation is conducted in four stages with a grinding step ahead of each of the first three stages (Fig. 6). Each successive regrinding mill, therefore, handles a lower tonnage of higher grade product. Various modifications of this flowsheet have been tried, but this one assures a satisfactory grade of concentrate with the least expense and loss in tailing

The first-stage regrind mills are charged with 1-in. steel and fine scrap from the 9 by 8-ft. mills. The mills on the final stages are charged with Danish flint. The reason for this is the bad effect of the by-products from steel in the subsequent flotation. This effect is greatly lessened if the pH in the mills is held at 8.2 to 8.4, but even so steel balls have a bad effect on recovery. Actually, flint has been found to be cheaper than steel, especially as the molybdenite content of the mill feed increases and the gangue decreases.

Fine grinding of molybdenite does not reduce its floatability if pebbles are used. Laboratory work on Climax concentrate has shown that grinding to minus 400 mesh in a porcelain mill charged with pebbles has had no detrimental effect on its flotation, which is not true if steel is used in the same manner.

The pH in the entire cleaning operation is maintained at 8.2 to 8.4 by the use of lime. The only other reagent is cyanide, which is added at various points in the circuit to depress the pyrite and chalcopyrite.

Approximately 99.5 per cent of the pyrite and 95 per cent of the chalcopyrite in the mill ore are finally eliminated. Greater elimination may be had with more intensive treatment with cyanide, but this is not required for the usual grade of product desired. The tailing from the cleaner plant is discarded. A screen analysis is shown in Table 1.

The final concentrate is filtered in Oliver filters, dried on Lowden driers and packed for shipment. Domestic shipments are packed in paper-lined burlap bags of 170-lb. capacity and foreign shipments are packed in 30-gal. oak barrels of 675-lb. capacity (Fig. 7).

Tailing Disposal and Water Reclamation

The tailing pond is on Robinson Flats, in Ten Mile Creek Canon, 2½ miles north of the mill and at a ground elevation 500 ft. lower.

Ten Mile Creek is a tributary of the Blue River, which in turn is a tributary of the Colorado River. The present site of the pond covers about 200 acres. Impounding in this pond was started in 1938.

The tailing pond serves the double purpose of storing and dewatering the tailing. No thickeners are used at the mill, as the slime in the tailing is slow settling on account of the low alkalinity of the water, pH 6.5 to 6.8, so that a prohibitive amount of thickening equipment would be required.

The tailing is transported by gravity from the mill to the pond in 18-in., inside diameter, machine-banded, Douglas fir stave pipe. The methods used for impound-

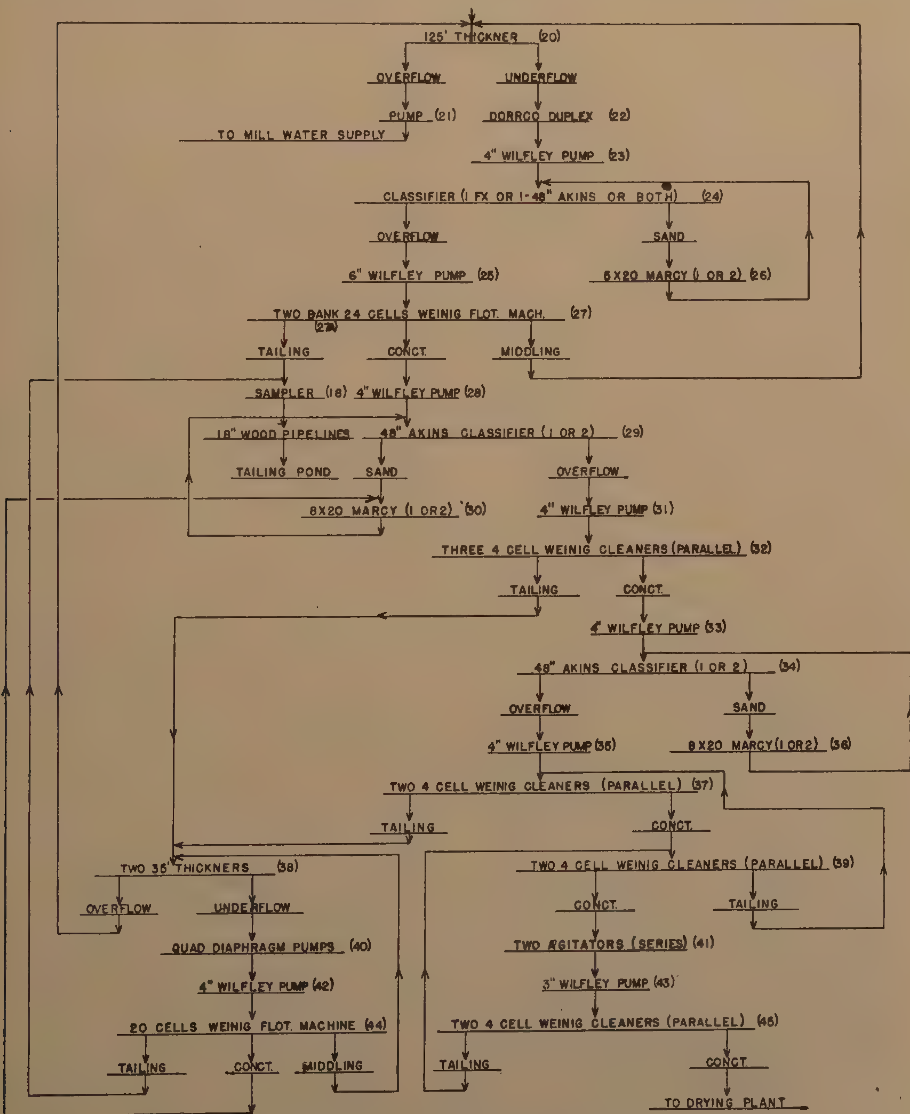


FIG. 6.—FLOWSHEET OF CLEANER PLANT, PRIMARY CONCENTRATOR.

ing are very similar to those used in the Southwest and elsewhere. Dikes of tailing are built around the outer edges by hand or machine and the tailing stream is directed to points within these dikes. The coarser material settles and most of the fine sands, slime and water, gravitate away from the dam toward the center. Three fourths of the water in the tailings is recovered in a clear overflow and returned to the mill.

In designing the present tailing-disposal system, it was considered desirable to use pipes for transporting the tailing instead of the launders used on the earlier ponds, chiefly because of economy of grade. The nature of the pulp to be handled created problems that are not usually encountered by other users of pipe lines for the same purpose. The pulp is not thickened and, although the solid content is high, it is less viscous than the usual thickener discharge on account of its relative coarseness. The slime is partly dispersed, therefore the pulp viscosity is lessened and the sands tend to settle out rapidly. The tailing may be as coarse as 20 mesh and a high velocity is necessary to keep a pulp of this nature in motion. The effect of high velocity, size of sand and abrasive nature of the sand is to create a condition in which abrasion is unusually severe.

Preliminary tests in small pipe lines indicated that a grade of 1.2 per cent would be the maximum required under any conditions and a grade of 1.5 per cent was selected as affording the necessary safety factor.

In order to keep the velocity at a minimum, the pipe was laid in a series of nearly horizontal sections, each succeeding section leading from the mill being at a lower elevation. The tailing is conducted from each horizontal section to that succeeding it by means of a vertical penstock open at the top, which automatically imposes the required head on the stream of pulp in that part of the line. Some of the "horizontal" sections are laid at 0 grade, some at 1 ft. per 1000 and some at 3 ft. per 1000. The mean hydraulic gradient from the top of any penstock to the downstream end of

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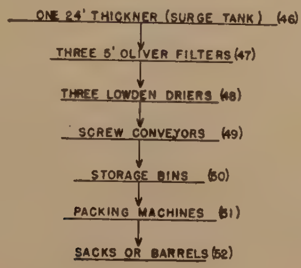
FIG. 6.—(*Continued.*)

(20) 125' DORR THICKNER - TORQUE TYPE - 0.099 RPM - 5 HP. GEAR MOTOR.
(21) ALLIS CHALMERS 6X6 CENTRIFUGAL PUMPS (2) - TYPE SF - 1070 GALLONS PER MINUTE - 40 HP. - 1750 RPM.
(22) TWO DORRCO NO. 6 V-TYPE DIAPHRAGM PUMPS - 3 HP - 865 RPM.
(23) TWO 4" WILFLEY PUMPS - 15 HP. - 875 RPM - DIRECT CONNECTED.
(24) ONE FX DORR CLASSIFIER 8'X27' 5 HP. 10 SPM. - ONE 48" AKINS DUPLEX SUBMERGED SPIRAL CLASSIFIER - 5 HP. - 5 HP. - SLOPE 2.75'/FT. - 1.5 RPM
(25) TWO 6" WILFLEY PUMPS - MOTOR - 30 HP. - 1165 RPM.
(26) TWO 5X20 MARCY TUBE MILLS - 25 RPM - 200 HP.
(27) FOUR 12 CELL 36" WEINIG FLOTATION MACHNES - 21" IMPELLOR - 300 RPM - 5 HP.
(27A) TWO 14X36 ROOTS CONNERSVILLE BLOWERS - 338 RPM - 40 HP - 2.2 LBS PRESSURE.
(28) TWO 4" WILFLEY PUMPS - 875 RPM - 15 HP.
(29) 48" AKINS DUPLEX SUBMERGED SPIRAL CLASSIFIER - 5 HP. - SLOPE 2.75'/FT. - 1.5 RPM.
(30) 8X20 MARCY TUBE MILL - 16 RPM - 200 HP.
(31) 4" WILFLEY PUMP - 875 RPM - 15 HP.
(32) THREE 4 CELL 36" WEINIG FLOTATION MACHINES - 21" IMPELLOR - 300 RPM - 5 HP.
(33) TWO 4" WILFLEY PUMPS - 875 RPM - 15 HP.
(34) 48" AKINS DUPLEX SUBMERGED SPIRAL CLASSIFIER - 5 HP - 2.75'/FT. SLOPE - 1.5 RPM. (THREE CLASSIFIERS AVAILABLE FOR (29) & (33))
(35) 4" WILFLEY PUMP - 875 RPM. - 15 HP (3 PUMPS AVAILABLE FOR 31 & 35)
(36) TWO 8X20 MARCY TUBE MILLS - 21 RPM - 200 HP. (THREE TUBE MILLS AVAILABLE FOR (29) & (35))
(37) TWO 4 CELL 36" WEINIG FLOTATION MACHINES - 21" IMPELLOR - 300 RPM - 5 HP.
(38) ONE 35' DENVER TRAY THICKNER - 9.2 REV PER HOUR - 3 HP. - ONE 35' DENVER SIMPLE THICKNER - 9.2 REV. PER HOUR - 2 HP.
(39) TWO 4 CELL 36" WEINIG FLOTATION MACHINES - 21" IMPELLOR - 300 RPM - 5 HP.
(40) ONE DENVER QUAD DIAPHRAGM PUMP - 56.5 RPM - 5 HP. - ONE DENVER DUPLEX DIAPHRAGM - 3 HP.
(41) TWO 8' STEEL TANK AGITATORS (CIRCULATION BY IMPELLOR AT BASE OF WELL) 3 HP MOTOR.
(42) TWO 4" WILFLEY PUMPS - 875 RPM - 15 HP.
(43) TWO 3" WILFLEY PUMPS - 1160 RPM - 10 HP.
(44) ONE 12 CELL 36" WEINIG FLOTATION MACHINE - 21" IMPELLOR - 300 RPM - 5 HP.
ONE 8 CELL 36" WEINIG FLOTATION MACHINE - 21" IMPELLOR - 300 RPM - 5 HP.
(45) TWO 4 CELL 36" WEINIG FLOTATION MACHINES - 21" IMPELLOR - 300 RPM - 5 HP.

its section of pipe is 1.5 per cent or greater. The length of these sections and the corresponding height of the penstocks vary to suit the topography. The longest stretch



(46) ONE 24' THICKNER (OPERATED AS SURGE TANK)
(47) TWO 5' 4" X 8' 0" OLIVER FILTERS - ONE 5' 4" X 10' 0" OLIVER FILTER.
(48) TWO 12' X 28' LOWDEN DRIERS (COAL FIRED) - ONE 12' X 48' LOWDEN DRIER (COAL FIRED)
(49) 15" SCREW CONVEYOR FROM EACH DRIER DISCHARGE TO JUNCTION POINT, THENCE BY 15" SCREW CONVEYOR TO STORAGE BINS.
(50) THREE 40 TON CAPACITY STEEL BINS.
(51) THREE ALLIS CHALMERS FLOUR PACKERS.
(52) SACKS (170 LBS. CONCT.) OR BARRELS (700 LBS. CONCT.)

FIG. 7.—FLOWSHEET OF FILTERING, DRYING AND PACKING PLANT, PRIMARY CONCENTRATOR.

of pipe is 3094 ft., laid on a grade of 3 ft. per 1000. The penstock feeding it is 42 ft. high, which gives a total head of 51 ft., or 5 ft. more than the required total of 15 ft. per 1000.

This method of laying the line ensures the minimum velocity and consequent minimum abrasion in the wood pipe. The pulp level rises in the penstocks so that sufficient head and no more is imposed to cause the pulp to flow. Normally the penstocks run approximately half full, so that more head is available if it should be required.

Another factor enters also to reduce abrasion. There is always a bed of coarse sand, which settles from the pulp and protects the bottom of the pipe, where most abrasion would occur if all the pulp were in motion. This bed of sand increases or decreases with the volume of pulp and the fineness of the tailing. That the open cross section of the pipe varies under different conditions has been confirmed by comparing the velocity with the pulp volume.

The difference in wear in pipe laid in this way and pipe laid otherwise has been amply demonstrated. It was necessary to lay a small part of the line on a sustained grade, and 16 ft. per 1000 was the grade employed. The wear in this part of the line has been many times as great as that in the flat pipes.

The penstocks, 4 by 5 ft. inside, are of considerably greater cross section than the conduits. This is to permit the expulsion of air, which in a smaller section would be entrained in the pulp stream and carried into the line. The greater section also lessens the turbulence and wear. Various types of penstocks have been tried out, the most practical being of monolithic concrete.

The open penstocks and horizontal lines afford an excellent opportunity to study the flow of pulps in pipe lines. Some of the results of these studies are given in Table 4.

Since the wood line was installed, experiments have been conducted with salt-glazed clay and concrete tile. Either of these has much greater resistance to abrasion than fir and the cost at Climax compares favorably with fir. The disadvantages of salt-glazed tile are that it is fragile and cannot stand much internal pressure.

TABLE 1.—*Typical Screen Analyses*

CRUSHER PRODUCT AND ROUGHER TAILING SCREEN SIZES

Size	Crusher Product (Feed to Primary Concentrator)		Rougher Tailing		
	Per Cent	Cum. Per Cent	Per Cent	Cum. Per Cent	Per Cent MoS_2
On					
3/8	6.9	6.9			
3	18.1	25.0			
4	16.9	41.9			
10	24.3	66.2			
28			1.4	1.4	0.300
35	13.7	79.9	4.6	6.0	0.209
48	2.3	82.2	9.5	15.5	0.124
65	2.0	84.2	13.0	28.5	0.069
100	2.0	86.2	11.7	40.2	0.044
150			11.6	51.8	0.032
200	3.6	89.8	9.6	61.4	0.022
Through 200	10.2	100.0	38.6	100.0	0.052

78-INCH AKINS DUPLEX CLASSIFIER PRODUCTS

Size	Ball-mill Discharge		Classifier Sand		Classifier Overflow	
	Per Cent	Cum. Per Cent	Per Cent	Cum. Per Cent	Per Cent	Cum. Per Cent
On						
20	31.7	31.7	41.7	41.7	0.1	0.1
28	10.2	41.9	12.7	54.4	1.0	1.1
35	11.4	53.3	12.0	66.4	3.9	5.0
48	9.7	63.0	9.4	75.8	9.6	14.6
65	8.3	71.3	6.5	82.3	13.5	28.1
100	5.1	76.4	4.2	86.5	11.0	39.1
150	4.7	81.1	3.0	89.5	10.3	49.4
200	3.4	84.5	1.8	91.3	7.7	57.1
Through 200	15.5	100.0	8.7	100.0	42.9	100.0

RE-TREATMENT SCREEN SIZES

Mesh	Rougher Concentrate		First Stage Overflow		Second Stage Overflow		Third Stage Overflow		Cleaner Tailing		
	Per Cent	Cum. Per Cent	Per Cent	Cum. Per Cent	Per Cent	Cum. Per Cent	Per Cent	Cum. Per Cent	Per Cent	Cum. Per Cent	Per Cent MoS_2
On											
65	14.0	14.0	1.0	1.0					0.3	0.3	
100	11.8	25.8	2.6	3.6					0.6	0.9	
150	10.0	35.8	6.3	9.9					2.4	3.3	
200	8.7	44.5	11.4	21.3	9.0	9.0	5.0	5.0	6.7	10.0	0.210
Through 200	55.5	100.0	78.7	100.0					90.0	100.0	0.090
On 325					17.0	26.0	8.5	13.5			
Through 325					74.0	100.0	86.5	100.0			

It would serve very well for a line laid on sustained grade, where the pulp velocity would be great enough to cause little or no internal pressure. Future tailing lines at Climax probably will be of one of these materials.

TABLE 2.—*Operating Data*

Tons per jaw-crusher hour	482
Tons per 7-ft. standard-cone hour	458
Tons per 7-ft. short-head cone hour (includes circulating load)	489
Tons per 7-ft. short-head cone hour (final product)	176
Tons per 9 by 8-ft. ball-mill hour	80
Manganese-liner consumption, jaws and Symons cones, lb. per ton	0.18
Chromium-molybdenum ball-mill liners, lb. per ton	0.201
Balls: 3-in. steel, lb. per ton	1.10
Water, gal. per ton	267
Pine oil, lb. per ton	0.20
Hydrocarbon, lb. per ton	0.60
Cyanide, lb. per ton	0.03
Lime, lb. per ton	0.07
Tons per man-shift crushing and milling	79
POWER CONSUMPTION	
Coarse crushing	1.91
Fine grinding	5.30
Flotation	1.80
Classifying	0.18
Pumping	0.25
Water	0.70
Concentrate treatment	1.19
Miscellaneous	0.23
Total	11.65

Two 18-in. lines run to the pond, either one of which will carry the entire stream. One of the lines is 16,700 ft. long and the other is 15,200 ft. They run along the east and west sides of Ten Mile Creek. They diverge as they leave the mill, one leading to the east side of the pond and the other to the west. By this means the tailing can be directed to any point that may be necessary.

The drain lines for decanting off the water settled from the tailing and conducting it to the pumping plant are of 24-in. and 30-in. lock-joint precast concrete tile. Connected to them at intervals are vertical monolithic concrete standpipes. The latter have slotted openings on one side, in which are placed concrete flash boards by means of which the proper water level is maintained. The pond site is on sloping ground and the decanting system is so arranged that as the height of the impounded tailing rises, decanters at higher levels will come into service and the lower ones may be blocked off. Ultimately, the tailing will have a depth of 300 ft.

TABLE 3.—*Effect of Grinding on Freeing Mineral from Gangue*

Mesh Through	Mineral Freed, Per Cent
CRUDE ORE	
28 (40 per cent plus 100)	40
35 (32 per cent plus 100)	60
65 (12 per cent plus 100)	70
ROUGHER CONCENTRATE	
200-mesh	90
325-mesh	95

Drain tile was laid under the concrete lines, under the toe dams and to various swampy spots as required. The ground was wet in places and drainage of ground waters was considered essential.

TABLE 4.—*Tailing-line Flow Data*

18-inch (inside diameter) wood-stave pipe
Length: 1900 ft. on grade 3 ft. per 1000 ft.
Fed by penstock 25 ft. high, 3 by 4-ft. inside.

Cu. Ft. per Min.	Solids, Per Cent	Dry Tons 24 Hr.	Plus 100-mesh, Per Cent	Head, Combined Line and Penstock, Ft. per 1000	Average Velocity,[a] Ft. per Min.	Average Open Section of Pipe[b]
412	34.3	7,890	20.0	6.95	304	77
402	39.4	9,480	25.6	7.35	310	76
406	41.4	10,340	31.3	7.63	304	76
404	44.0	11,080	34.8	7.81	304	75
409	49.4	12,920	40.8	8.95	304	76
405	51.1	13,740	43.4	9.20	310	74
206	34.3	3,945	20.0	6.32	262	46
201	39.4	4,740	25.6	6.95	262	44
203	41.4	5,170	31.3	7.63	253	45
202	44.0	5,540	34.8	8.25	260	42
205	49.4	6,460	40.8	9.05	258	44
202	51.1	6,870	43.4	10.00	262	44

[a] Velocity determined by adding salt to intake of penstock, analyzing pipe discharge for chlorine and noting time for salt to appear at discharge end.
[b] Part of pipe area required to carry volume at determined velocity.

The building of dams has been done mostly during the summer months. This will be comparatively simple, as a rise of 10 to 12 ft. per year will take care of the probable tonnage of tailing.

Robinson Lake, 3½ miles from the mill and 600 ft. lower in elevation, is the mill-water reservoir. It is a natural lake, the capacity of which was increased to 2000 acre-feet by a dirt fill dam. The dam has a crest length of 1341 ft. and a maximum height of 81 ft. The lake is filled by means of a system of canals, which gather the spring runoff from various areas of the Ten Mile Creek watershed, supplemented by water recovered from the tailing pond, as needed.

The pumping plant has a capacity of 4000 gal. per min. and delivers the water to the mill through 3½ miles of 20-in. welded steel pipe. The pumps and line are provided with the most modern equipment to guard against the shocks of starting, stopping and surges from any cause.

Safety

A comprehensive safety program under the supervision of an independent department is in force, including periodical safety meetings, contests, adequate guarding of equipment and, most important of all, constant education of the operating personnel.

Acknowledgments

Thanks are due to Mr. W. J. Coulter, Manager, and Mr. C. J. Abrams, Superintendent of the Climax Molybdenum Co., for permission to publish this paper; also, to Mr. O. E. Young and Mr. R. C. Cuthbertson, of the Metallurgical Staff, and Mr. Franklin Coolbaugh, Assistant Mill Superintendent, for assistance in its preparation.

DISCUSSION

(*Charles E. Locke presiding*)

R. P. Jarvis,* Toluca, Mexico.—On page 593, Mr. Duggan says, speaking of the speeds of drive for ball mills: "The mills have been driven at various speeds from 17 to 22 r.p.m. Allowing for wear of shell liner, this gives speeds from 66 to 84 per cent of critical. The higher speeds have yielded greater tonnages, but not in proportion. *As the power increases directly as the speed, the lower speeds are more efficient per unit of power.*" I believe the last statement to be in error, for the power increases not simply as the speed but as the *square* of the speed or velocity, which makes quite a difference, which easily accounts for his observation that "the lower speeds are more efficient per unit of power." With speeds of 17 and 22 r.p.m., the power requirements would be in the proportion of 289 to 484, or in speeding up to 22 r.p.m. from 17 r.p.m., all other conditions being the same, the increased power requirements would be 167 per cent at the higher speed.

E. J. Duggan (author's reply).—The following are the power requirements of our 9 by 9-ft. ball mills at various speeds. The kilowatt load is the input to the mill motors with the mills operating under as nearly constant conditions, with the exception of speed, as was possible under operating conditions. These readings were all taken before the mills were converted from high discharge to low discharge.

Speed, R.P.M.	Kilowatt Input
17.09	281.7
18.99	313.6
19 94	324.2
21.83	360.6

* Geologist and Mining Engineer.

Milling Practice of the St. Joseph Lead Company

By H. R. Stahl,* Member A.I.M.E.

(St. Louis Meeting, October 1942 and New York Meeting, February 1943)

The disseminated lead district of Southeast Missouri lies 70 miles south of St. Louis. The only metal of economic importance in the ore is lead, but minor amounts occur of iron, zinc, copper, cobalt, nickel, silver and cadmium. These minor metals, as might be expected, tend to concentrate in both the flotation and table middlings. In the past, some zinc has been recovered by differential flotation. The silver and cadmium tend to associate with the zinc, and some recovery is made of these two metals at the smelter.

The lead occurs as galena, or lead sulphide. The minor metals are also present as sulphides. The presence of oxidized lead is suspected, but has not been definitely proved. The gangue is chiefly dolomitic limestone, with small amounts of sandstone and shaly glauconitic and chloritic minerals. The ore contains about 3.25 per cent lead, occurring as galena disseminated through the gangue. Some of the galena is very finely disseminated, and the mineral in the gravity tailings, or chat, is seldom or never in the free state.

The method of treatment of the ore has been evolved over a period of 75 years; hence probably is not far from correct. The milling processes have tended toward simplification during the past 15 years, but recoveries have never been sacrificed for the sake of simplicity. In many cases, simplification of flowsheets has led to better recoveries and operating economies. A simplified flowsheet is shown in Fig. 1.

The St. Joseph Lead Co. operates four mills in the Flat River district, with a normal total capacity of about 21,000 tons per day, divided as follows: Federal mill, 11,000 tons; Leadwood mill, 4500 tons; Desloge mill, 3400 tons and Bonne Terre mill 2400 tons. The methods of concentration used at these plants are similar in principle; they differ mainly in details and in the type and size of the equipment used. The process of concentration may be divided into five main steps; i.e., dry crushing, wet grinding, classification with its attendant desliming, tabling and flotation. A brief description of these processes will be given, noting important differences in the various plants, and also noting the more important changes or developments of the past few years.

Dry Crushing

The run-of-mine ore is hoisted in skips, the capacity of these skips ranging from 2¾ tons to 8¼ tons. At the Desloge plant, the ore is hoisted in 2-ton mine cars on self-dumping cages. All ore is hoisted at shafts adjacent to the various crushing plants, no ore being transported on the surface. Primary breaking at all the mills is accomplished with 9E Telsmith breakers. Preliminary screening is provided ahead of most of these crushers, either by station-

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ary or revolving drum-type grizzlies, eliminating as much as 40 per cent of the ore hoisted in some places. As these crushers have a relatively close setting, sometimes delivering a maximum product of minus 3-in., this by-passing of fine material has proved very beneficial in reducing the load on the crushers.

The Telsmith crusher product is conveyed to grizzlies, either of the stationary-bar type or vibrating-screen type, where a separation is made at about one inch. The grizzly oversize is fed to secondary crushers, the various mills using different types, such as No. 6 Telsmith, 5K Gates, and No. 5 McCully. These crushers, in general, reduce the Telsmith breaker product to a material containing about 50 per cent minus ½-in. and 5 per cent plus 1 inch.

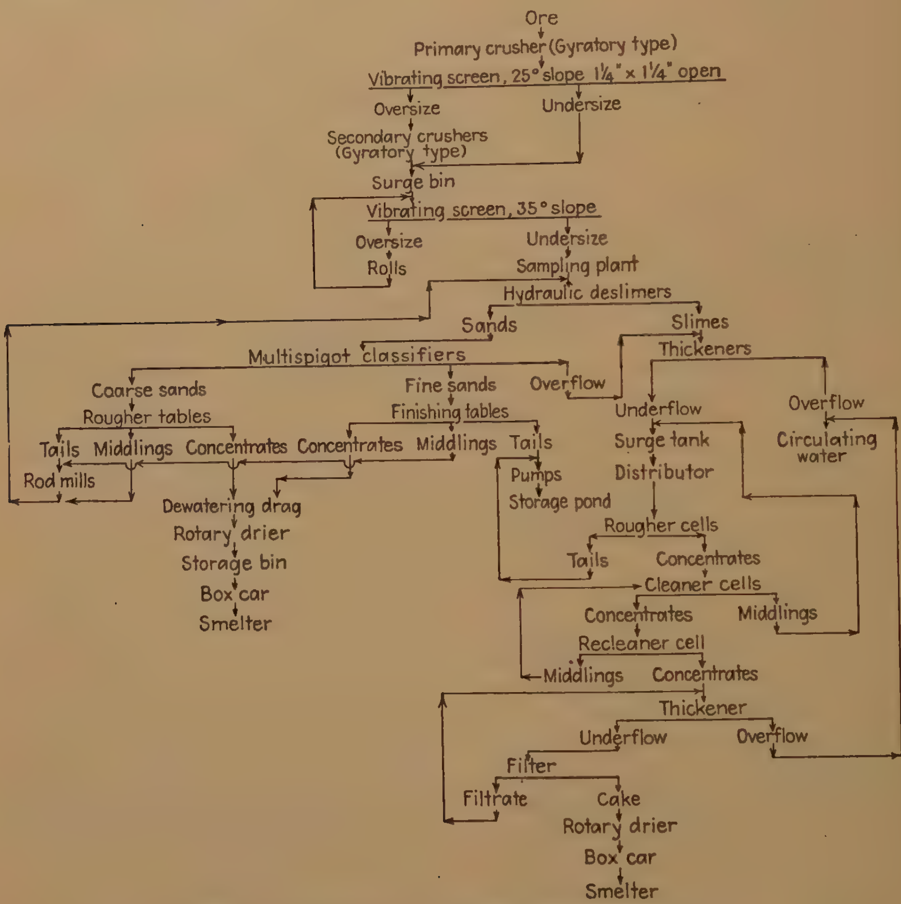

FIG. 1.—SIMPLIFIED FLOWSHEET, BONNE TERRE MILL.

The secondary crusher product joins the grizzly undersize and is conveyed either to storage bins or directly to dry screens. Storage bins at some point in the dry-crushing circuit, at all the mills, either precede or follow the screens. The storage bins are large except at one place, where the capacity is only about 100 tons. The largest bin, that of the Federal mill, has a capacity of about 4000 tons. These bins are a great aid in equalizing irregularities

in hoisting, crushing and milling, permitting more continuous operation of the entire mining-milling system. The large bins are all equipped with automatic ore trippers. All ore is weighed on Merrick weightometers, at some point in the dry-crushing system.

The product from the second stage of dry crushing is screened through vibrating screens, using wire-mesh cloth, the openings being usually close to 0.100 in. Square-mesh cloth is used at some places, while the Rek-Tang type of cloth is preferred in others, as it seems to handle wet, muddy ore better. A backing sheet of heavier wire, with 1-in. openings, is used beneath the top sheet. Four types of vibrating mechanisms are used in the district, three being of local design.

The dry screen oversize constitutes the roll feed. Rolls are of various sizes and types, but all are of the standard belted type, using a countershaft between the driving motor and the roll. Draw rods hold the driven roll bearing against shims, so that most of the rolls almost touch each other. Except for some of the larger rolls, one shell is not belted, and revolves by friction against the driven shell when ore is being crushed. The old Cornish type of geared roll was in use in the district until comparatively recent times—at one mill as late as 1933. The types of rolls used are Traylor, Allis-Chalmers and Buchanan, and vary in size from 36-in. to 64-in. dia., and in width of face from 15 to 26 inches.

The roll product is returned to the screens, thus closing the dry-crushing circuit. The circulating load in the roll-screen circuit is large, especially with damp ore, and amounts to 5 or 6 to 1 in most cases. The tendency in recent years has been to carry the dry crushing to a point where about 90 per cent of the product passes 14 mesh, as experience has shown that this liberates the larger part of the galena and a smaller percentage of the ore must be reground in rod mills. Less colloidal or finely divided mineral is produced by dry reduction, enabling the plants to recover a larger proportion of the mineral as gravity concentrates.

Classification and Desliming

The screen undersize, after being mechanically sampled, is deslimed and classified. The desliming is accomplished in Delano hydraulic deslimers, which are essentially cones, having a top diameter of 80 in., a central pulp distributor with a baffled bottom to break the force of the incoming stream, and a bottom chamber having a clear-water inlet and a spigot for continuous discharge of the sands. The slimes overflow in a launder surrounding the upper edge. The spigot discharge is a mixture of sands of many sizes. In order to treat these sands effectively on tables, a sizing or classification is necessary.

The classifiers used are a local modification of the Fahrenwald design. The machine consists of a series of cells 14 by 14 in., each cell having a perforated plate in the bottom, through which hydraulic water is introduced. A discharge opening in the center of each plate, fitted with a removable spigot, discharges the sand from each cell. All slimy material not previously removed by the Delano deslimer overflows into a launder along the side or end of the classifier. This overflow, together with the overflow of the Delano deslimers, is laundered to Dorr thickeners for settling. The classifiers at the various mills are of different sizes, ranging from 13 to 28 cells in length. In some places, to obtain added capacity without undue length, cells are placed side to side; i.e., a 30-cell classifier may consist of two adjoining rows of 15 cells each. By careful adjustment of spigot openings and of the hydraulic water, a well-graded series of sands is obtained. These sands constitute the feed to the various tables.

Tabling

Each table makes three products—a finished concentrate, a middling that is either recirculated through the deslimer and classifier or sent to the rod mills or ball mills for regrinding, and a tailing that is either reground or sent to waste. The rougher, or coarsest tables, which vary in number according to local conditions, make a middling that may contain considerable free mineral at times. This is sent back to the deslimers and classifiers for recirculation. The tailings from these rougher tables is sent to mills for regrinding. The finishing, or finer tables, make a middling of locked particles, which is returned to the mills for regrinding. The tailings of the finishing tables are sent to waste.

Rubber covers, 5/32 in. thick, are used for table-deck coverings. This type of cover has entirely replaced the linoleum or concrete covers of 15 or 20 years ago. The type of rubber cover now used will give from 10 to 15 years of service. Riffles are of molded rubber and are laid parallel to the axis of motion.

At one mill in the district, Leadwood, the tabling procedure differs from that just described. At that mill table middlings are reground separately from the rougher table tailings and are treated on a separate group of tables.

Dry Crushing Replacing Wet Screening

In recent years, finer dry crushing and classification have replaced wet screening. Formerly, in order to maintain a satisfactory grade of concentrate and tailing, it was necessary to wet-screen through about 10 mesh, sending the wet-screen oversize to rod or ball mills for grinding. Under the classification scheme, the classifiers send a feed to the tables from which as much coarse material as desired may be sent back to rod or ball mills and again to the classification system. Formerly the ball or rod mills were in closed circuit with wet screens; now they are in closed circuit with classifiers and tables. The classifier system is quite flexible in operation. For instance, in a battery of 10 tables, the tailings of the first 4 may be reground, and the tailings of the last 6 sent to waste. Should the tailing of the fifth table run too high in lead, it may be sent to regrind. In other words, the cut-off point between waste tailing and regrind tailing can be changed at will to meet conditions. Some of the mills still use wet screens in their circuits, but at present these screens perform no sizing duty and are used only to trap oversize rock and foreign material.

The table middlings and coarse tailings from the rougher tables, as stated, are ground in rod or ball mills. The size and type of mills used vary throughout the district. Two of the plants are using 6 by 4-ft. mills, originally ball mills but now using 1¾-in. rods, 51 in. long, as grinding media. One mill uses 4 by 10-ft. mills, originally rod mills but now charged with 2-in. balls. The fourth plant uses 6 by 12-ft. rod mills. These mills are all of the open trunnion-discharge type and lined with smooth 2-in. liners of manganese steel, or chrome-nickel-iron. Usually the rod-mill or ball-mill feed is not dewatered. The table rejects are sluiced to a box at the feed end of the mill, whence the material enters the mill through a pipe feeder. The pulp density in the mill therefore is low, ranging from 30 to 40 per cent solids. This comparatively large volume of dilute pulp minimizes the production of slime. Formerly the feed to the mills was dewatered by dewatering wheels or Esperanza drags, and introduced into the mills by scoop feeders, where a pulp density of 65 to 70 per cent solids was maintained. The adoption of the newer scheme of more dilute grinding, besides decreasing slime production, has made possible the elimination of the accessory dewatering apparatus mentioned above, thus decreasing main-

tenance costs. The reground material from the ball or rod mills is pumped or elevated back to the deslimers and rejoins the classifier-table circuit.

Thickening

The slimes from the deslimers and classifiers are laundered to Dorr thickeners. Various sizes of thickeners are in use, ranging in diameter from 40 to 150 ft. The clear water overflowing the thickeners is returned to the mill circulating-water system for re-use. The spigot product from the thickeners, at 30 to 35 per cent solids, is distributed to flotation machines, each unit consisting of a rougher cell 36 ft. long and a cleaner cell 12 ft. long, equipped with an individual centrifugal air compressor, delivering 4000 cu. ft. of free air per minute at ¾ lb. pressure. The rougher cells make a final tailing ranging between 0.08 and 0.15 per cent lead. The froth from each rougher gravitates to the cleaner cell in each unit. The cleaner-cell middling returns to the original feed to the roughers, making a circulating feed of about 24 per cent solids. The cleaner froth, assaying 15 to 18 per cent lead, is stepped up to 65 to 75 per cent lead by one or two re-cleanings, for a final concentrate. The percentage of lead in the concentrate depends in some cases upon the amount of zinc carried, which often runs from 5 to 10 per cent in certain localities. The presence of zinc in the concentrate is not objectionable, as some silver and cadmium are associated with the zinc and a recovery of these two metals is made at the smelter.

Flotation

Flotation reagents vary somewhat at the different plants, but a mixture of ¾ hardwood creosote and ¼ Sharples' Pentasol frother is generally used as a frothing agent. Three of the mills use Aerofloat No. 31 as a collecting agent, while the fourth has found Sodium Isopropyl Xanthate the most satisfactory. Reagent costs are low, being about 2 to 3 cents per ton treated.

The quantity of the total mill tonnage treated by flotation averages 61 per cent for all four mills in the district. However, the flotation plants produce only about 40 per cent of the total concentrates, because the flotation process is really a scavenging, or clean-up, procedure. Considerable fine galena is recovered in the classifier-table circuits, leaving an impoverished feed for flotation treatment.

Flotation practice is almost uniform throughout the district. The Desloge mill varies the usual procedure by treating in a separate plant the slimes produced by dry crushing. Experiments proved that the original or primary slimes at this plant were much less responsive to treatment than the slimes produced by regrinding in the ball mills. This original slime is sulphidized with sodium sulphide and treated in a separate flotation circuit.

The final flotation concentrate is settled in Dorr thickeners. The thickener underflow, at 65 to 75 per cent solids, is sent to Oliver filters, the common size in use being 11 ft. 6 in. in diameter by 12 ft. long. The filter cake, containing about 15 per cent moisture, is dried in rotary driers, 3 ft. in diameter by 20 ft. long. The driers use natural gas for fuel. A hood at the end of the drier receives the concentrate, which carries 5 to 7 per cent moisture. The dried material is conveyed to box cars, where it is distributed by a mechanical box-car loader. The rotary gas-fired driers have replaced the coal-fired, hearth and rabble type formerly used in the district. A great saving in maintenance costs has been effected by their use. The gas pressure used varies from 5 to 27 lb., dependent upon the quantity and moisture of the filter cake. From 300 to 400 cu. ft. of gas is consumed per ton of concentrate dried, at a cost of about 9¢ per ton. The rotary drier discharges the concentrate in the form of small pellets, which make a dustless product and

one easily loaded and handled at the smelter. The box-car loader is a small but important innovation of recent years, as it dispensed with the slow and laborious work of loading by wheelbarrows.

Treatment of Concentrates

The gravity concentrates from the tables are handled by two different methods. At the Bonne Terre mill, the concentrates are dewatered in an Esperanza drag classifier to 14 to 15 per cent moisture and then dried in a 2 by 8-ft. rotary gas-fired drier to 2 to 3 per cent moisture. At the other mills, the table concentrates are pumped to storage cones, from which a thickened product is spigoted to Dorrco filters. Each filter is 8 ft. in diameter by 2 ft. in depth and delivers a product carrying 3.0 to 3.5 per cent moisture. The filtered concentrate is conveyed to box cars and loaded in a manner similar to that of the flotation concentrates. Filtering of the table concentrates is a development of recent years and has demonstrated its value. The concentrates formerly were dewatered in Esperanza drags and loaded in cars by wheelbarrow. The drip from the car caused loss of fine material and the moisture content of 7 to 8 per cent caused trouble in cold weather by freezing en route to the smelter. The filtered concentrates are sufficiently dry to prevent dripping and hard freezing.

Table Tailings

Table tailings, or chat, are dewatered by means of classifiers of the Esperanza or Dorr type, or by Delano deslimers. The overflow from the various dewatering devices, containing some small lead values, is sent to the thickeners or used as wash water in launders, or other apparatus. The dewatering tailings, usually combined with the tailings from the flotation plant, are pumped to storage ponds. Pipe lines are run along the ends of the ponds and some of the tailings are withdrawn at intervals along the lines in order to build up a dam above the overflow level in the pond. This method of tailings disposal has superseded the conveyor method of stacking the tailings on piles or dumps. After the elimination of Hancock jigs and the consequent finer grinding of recent years, the tailings produced were fine enough to be handled readily by pumps, and this method of waste disposal has proved more economical and satisfactory than conveying the material to dumps.

Upkeep of Equipment

Individual motor drives are largely used throughout all the mills. Virtually all the heavier drives, such as primary crushers and rolls, use endless belts. The replacement of spliced belts by endless belts a few years ago has been an important factor in obtaining smoother operation of the heavier equipment.

Wearing parts, such as roll shells and crusher mantles and concaves, are kept to shape by means of hard surfacing, thereby saving much time, labor and material formerly consumed in repairing or changing heavy parts. All heavy equipment is served by air hoists or electric cranes to facilitate prompt and easy handling during repairs or replacements.

Trend toward Simplification

Milling practice in the Lead Belt has never been stationary at any time during the 75 years of its history. Flowsheets are never fixed for any considerable length of time, and each of the four concentrating plants is a separate laboratory, in which new ideas and changes are constantly being introduced. While some of the changes in future years may not be as far-reaching as some of the changes in recent years, it is safe to predict that the trend toward simplification of processes will continue.

Ohio Copper Company Tailings Re-treatment Plant

By Frank R. Milliken,* Member A.I.M.E., and Robert Goodwin†

(Salt Lake City Meeting, September 1940)

In September 1937, the Ohio Copper Co. inaugurated the treatment of its copper-bearing mill tailings at Lark, Utah. These tailings had been accumulated during the regular operation of the Ohio Copper mine and mill over the period 1907 to 1919. Gravity concentration had been employed except for a comparatively short period in 1919, when flotation was used.

The ore milled during that period assayed about 1 per cent Cu. The chief copper mineral, chalcocite, was disseminated in a shattered quartzite, mainly along fracture or cleavage planes. The quartzite gangue was extremely hard and grinding to liberation resulted in excessive sliming of the "sooty" chalcocite, with consequent high tailings losses. Recovery on the seven million tons treated was probably less than 60 per cent of the total copper content.

As early as 1918, attempts had been made to sample the dump. In 1928, a complete sampling program was undertaken. The sampling consisted of seventy-four 5-in. post auger holes drilled to the full depth of the dump at regular intervals of 250 ft. Each 10-ft. portion thereof comprised a sample for analysis. Each sample was screened, rolled, divided into three parts, dried and assayed. The reject was retained for testing purposes. From the depth of the auger holes a contour map of the dump was plotted, Fig. 1, which gives percentages of tonnage and copper content. The tonnage within the limit of the drill holes was estimated at about five million tons with an average content of 0.42 per cent Cu, 31 per cent of which (0.13 per cent) was acid-soluble. A screen analysis of the composite dump sample is shown in Table 1.

Flotation tests by several companies and individuals were then made, the results of which ranged from 62 to 74 per cent recovery. On the basis of these, a 200-ton pilot plant employing straight flotation was built and operated for a few months.

Table 1.—*Screen Analysis of Dump*

Mesh	Percentage of Sample by Weight	Assay, Per Cent Total Copper	Percentage of Total Copper
On 65	17.12	0.34	12.27
On 100	12.71	0.22	5.89
On 150	19.62	0.22	9.10
On 200	17.12	0.22	7.93
Minus 200	33.43	0.92	64.80

Operation of this plant was unsuccessful, as no provision had been made to handle the copper-bearing solutions resulting from the water-soluble copper, a point that was overlooked in the laboratory test work. Attempts to decant the copper-bearing solution for precipitation in the usual manner ahead of flotation were not satisfactory because of low water recovery (owing to poor settling characteristics of the pulp) and contamination of the copper precipitate with unsettled slime. Low copper prices in 1931 necessitated the termination of testing.

With the stimulus of higher copper prices in 1937, a review was made of metallurgical

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NOTE: ALL CONTOURS ARE OF ORIGINAL GROUND

FIG. 1.—MAP OF TAILINGS DUMP, OHIO COPPER COMPANY, LARK, UTAH.

possibilities for the economic treatment of the dump. This showed:

1. Straight flotation, even with acid-resistant and copper-resistant equipment, could not be considered, as flotation recovered only 70 per cent of the sulphide copper in the ore, an over-all copper recovery of less than 55 per cent. Sulphide flotation was complicated by the presence of chalcanthite, brochantite, and other basic copper sulphates as well as by the rapid oxidation, even during treatment, of the sulphide minerals themselves. Grinding to increase sulphide recovery by further liberation or by producing fresh surfaces and brightening old ones was of no benefit.

2. Treatment of the dump for water and acid-soluble copper alone by leaching or decantation was not possible because of the low soluble content and the poor percolation and settling characteristics of the solids, particularly in the higher grade regions of the dump, naturally high in slimes.

3. These same poor settling characteristics also precluded the recovering of the soluble copper by decantation (with launder precipitation of the clear solution) and the sulphide copper by flotation. The capital and operating costs of such a flowsheet would be economically prohibitive.

In addition to the metallurgical characteristics of the ore, other factors that had to be considered were:

1. The material would be received at the mill in a pulp, as sluicing was the least costly way to handle this material.

2. Both capital cost of equipment and operating cost had to be low, to permit amortization of the capital investment and to allow a reasonable return.

3. Metallic iron and acid were available at relatively low prices and smelters were near by.

Considering all these conditions, the only treatment economically sound was the leach-precipitation-flotation process, involving leaching of the water-soluble and acid-soluble copper, precipitation of this soluble copper on metallic iron in the pulp, followed by the simultaneous flotation of the cement copper precipitates and the sulphide copper.

A somewhat similar process is being successfully used by the Miami Copper Co. in treating a so-called mixed ore, but essential differences in the two ores required modification of the process in equipment and operation at the Ohio Copper Company's plant. First, the sulphide and cement copper were to be floated together, necessitating the use of a circuit to give the best results on the mixture rather than two separate circuits, as at Miami, where each circuit is adjusted for one type of flotation. Second, the solution for precipitation was so low in copper content that an inexpensive precipitator would be required, which could be cheaply operated. The flowsheet finally adopted for the treatment of 1000 tons of tailings per day is shown in Fig. 2.

Sluicing

The dump covers an area roughly 2000 by 3000 ft. and is from a few feet to a maximum of 55 ft. deep. There are two distinct draws or arroyas underneath the tailings, with the general slope toward the mill. However, there is insufficient grade to deliver the pulp to the mill by gravity in launders (grade required is 2.5 per cent), so the pulp is sluiced to a sump from which it is pumped to the mill.

The deposit is a series of ponds whose banks were built with the coarse sand and in whose centers the fines or slimes was settled. Other ponds were deposited on top of the first, so that in some sections of the dump alternate strata of sand and slime occur. Where this condition exists, the sluicing of a good mixture is simplified. In other sections the material is sandy or slimy from top to bottom and with this condition it is often necessary to have one monitor working in the sand and the other in the slime at a considerable distance from each other.

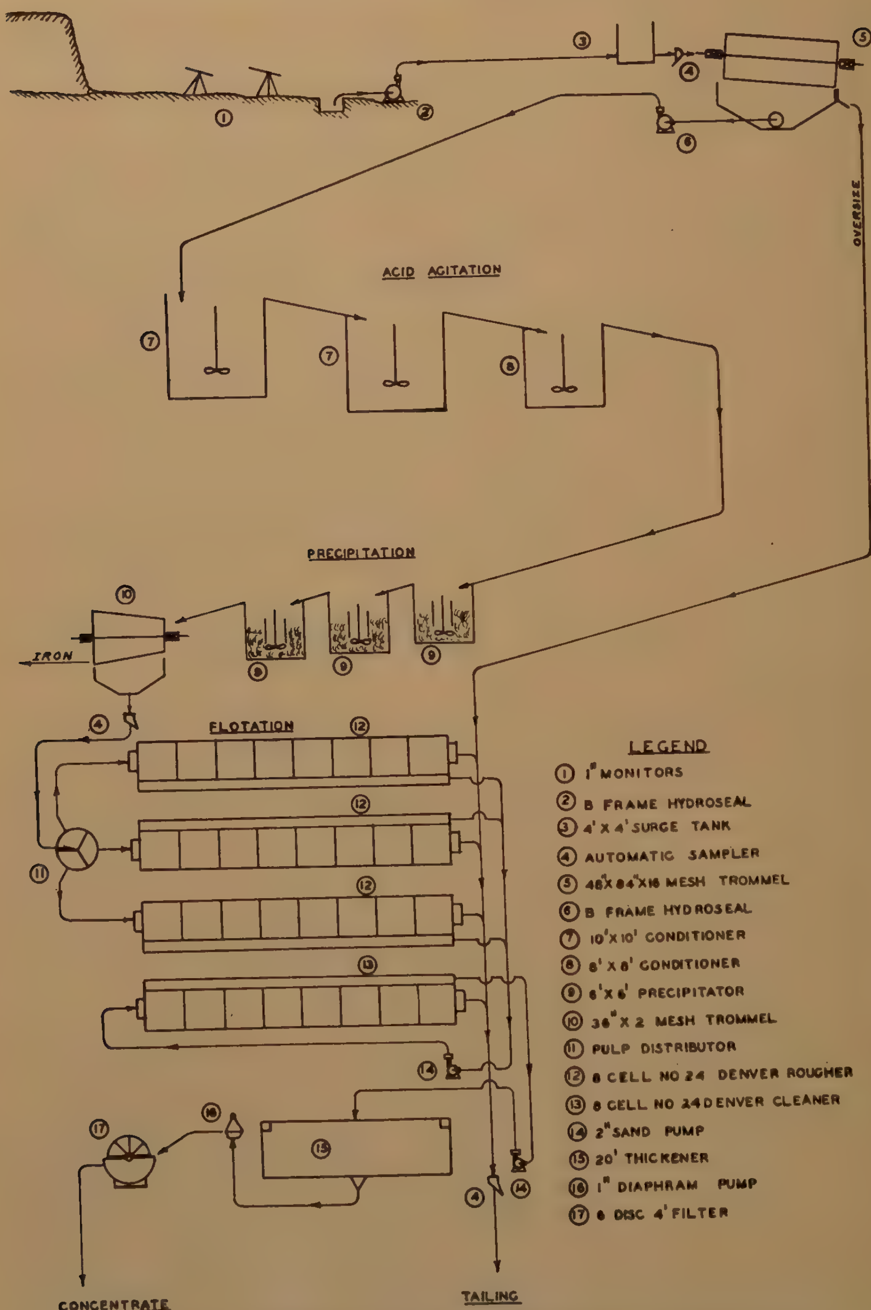

FIG. 2.—FLOWSHEET, OHIO COPPER COMPANY'S TAILINGS PLANT.

Mine water from the Mascotte tunnel flows through a wooden launder across the upper part of the dump to the pressure pumps. These are two Ingersoll-Rand 15-hp. pumps, which deliver 400 gal. per min. through a 6-in. tarred iron pipe to the two monitors. The pressure at the monitors is about 90 lb. per sq. in. Nozzles of various sizes have been used, but the ⅞-in. nozzle is best suited to the purpose. The sluicing water is slightly acid and contains soluble salts of iron, aluminum, etc., but is not highly corrosive to iron pipe. The pipe from pressure pumps to monitors has been in service over two years with little replacement.

An attempt is made to hold the mill feed as close to the average copper content of the dump as possible. There are several reasons for this. First, the coarse sand, or low-grade material, will not flow in the launders without being mixed with an equal amount of slime. The advantage to the mill of a uniform grade of feed is obvious but an added advantage of mixing sand and slime is apparent in the mill operation. The sand appears to have a beneficial effect in conditioning the minerals in the higher grade slime.

The density of the mill feed is controlled entirely by the monitor operators. An electric vibratory horn system is installed between the monitors and the mill to enable the mill operator to keep the monitor operator advised of the percentage of solids in the pulp. A density of 30 per cent solids is maintained; the half-hourly determinations do not ordinarily vary much from this figure.

In order to prevent loss of soluble copper by seepage, launders and monitors are kept as close to the sluicing face as safety will permit. The launders deliver the pulp to a wooden sump, where it is pumped to the mill through a 5½-in. wood pipe by a rubber-lined 6-in. B-frame Hydroseal pump. A wooden grid with a 1-in. opening ahead of this pump screens out brush, scrap wood and rock, of which there is a considerable amount.

Daily tonnage is estimated from the known factors, size of nozzle, nozzle pressure, and percentage of solids in the pulp.

Routine labor in the dump consists of occasional addition of pipe to the pressure line, addition of new launders and frequent moving of the monitors to obtain maximum efficiency of the sluicing operation. Ordinary fire hose is used for a flexible connection between the pressure pipe line and the monitors. Costs of sluicing in the winter months are higher than during the remainder of the year because of the lower tonnage handled. Water from the nozzles freezes on the sand and slimes faces and there is a marked reduction in the efficiency of the operators in very cold weather.

During 1938 and 1939 a corridor was cut almost through the length of the dump. Because of lack of sufficient launder grade, this corridor was not cut from top to the bottom of the deposit, but the average grade of the mill feed during this period checked very closely with the dump sampling.

Milling

The pulp from the sluicing operation is delivered at approximately 30 per cent solids into the bottom of a surge tank, from which an orifice outlet provides a steady feed to the plant.

The surge-tank discharge is sampled automatically to obtain heads to the mill, and is then discharged into a 48 by 84-in cylindrical trommel equipped with a 16-mesh stainless-steel screen. The trommel frame is fabricated of Everdur, rubber, and stainless steel. The trommel acts primarily as a scalping screen, removing debris and rock particles coarser than 16-mesh, which would otherwise collect in the precipitators. At times, some slime balls not broken up in the pump or pipe line are also discarded. This could be avoided by placing the screen after the acid agitators, but the

coarse material would then go through these agitators, giving some mechanical trouble.

The screen undersize is pumped up to the acid agitators by a B-frame Hydroseal rubber-lined pump. The acid agitators, of the Devereaux type, provide about 30 min. contact time. The tanks are wood, rubber lined, and the ship-type impellers are rubber covered. The acid agitation section serves four purposes:

1. Additional solution of copper. About 25 per cent of the copper in the ore is water-soluble and acid-soluble, with about half of this amount water-soluble. Of the feed to the plant, then, about 12.5 per cent of the total copper is in solution and after acid agitation about 25 per cent is in solution.

2. Cleaning of sulphides. The acid leaching removes oxide films from the tarnished sulphide mineral surfaces, increasing materially the floatability of the sulphides. Without this acid treatment, sulphide flotation is unsatisfactory.

3. Conditioning with Minerec. The contact time provided gives ample opportunity for conditioning the sulphides with Minerec (0.025 lb. per ton ore).

4. Preventing surges to flotation. Sudden fluctuations in the density and character of feed from the dump are leveled out to a large extent, permitting effective control of the flotation circuit.

On an average, about 6 lb. of 60° Bé sulphuric acid, all added to the first agitator, is used per ton of ore, the acid costing $6 per ton at the mill (a cost of 1.8¢ per ton of ore). About one pound of copper per ton of ore is actually dissolved by this acid. However, it is not fair to charge the entire acid cost against the copper dissolved, as the acid is necessary for satisfactory sulphide flotation and would be used in any case.

A free acidity at the end of agitation of 0.5 to 1.0 lb. sulphuric acid per ton of solution is maintained, depending on the character of the ore. As the "slime" content increases, a higher final acidity is required for satisfactory results. Gangue consumption of acid rarely exceeds 3 or 4 lb. per ton of ore.

The discharge from the acid agitation section goes directly to precipitation. Three 6 by 6-ft. rubber-lined wood-tank agitators in series serve as precipitators. Thirty-inch four-blade ship-type impellers, each driven at 250 r.p.m. by a 15-hp. motor, provide the agitation.

A well, 32 in. inside diameter, provides circulation of the iron and pulp and protects the impeller during shutdowns. The impeller, of white iron, lasts about 20 days. This excessive wear is due to the combined effects of corrosion and abrasion, but as the replacement cost is low more expensive corrosion-resistant material has not been substituted.

The iron used for precipitation is purchased from the Los Angeles By-Products Co. and is known as "Premt" or "shredded iron." The bulk of the iron, obtained from discarded tin cans, is shredded into pieces about 2 in. square. Shredding is accomplished in a swing hammer mill, replacing the hammer with heavy knives.

The iron is added by hand to maintain a predetermined agitation condition in each tank. For effective operation with the present mechanical setup, about 0.6 tons of shredded iron is maintained in the precipitators per ton of solids (a ratio of iron to solution by weight of 1.0 to 4.0). A higher precipitation rate and lower final precipitation tailing can be obtained by increasing the ratio of iron to solution up to 1.0 to 2.0, but at the present time the precipitation loss is only 0.29 lb. copper per ton of ore, and other more important factors have been receiving attention.

The three 6 by 6-ft. precipitators provide about 5-min. contact time on the regular flow of pulp, which is sufficient, for the iron concentration used, to give 82.9 per cent precipitation of the copper in solution. In

Fig. 3 are shown time-precipitation curves for the Ohio Copper solution for various ratios of iron to solution. In addition, laboratory results obtained on other grades of solutions are shown. Regardless of the grade or head value of the solution treated, with a ratio of iron to solution of 1.0 to 2.0, the economic limit of precipitation is reached in about 5-min. contact time.

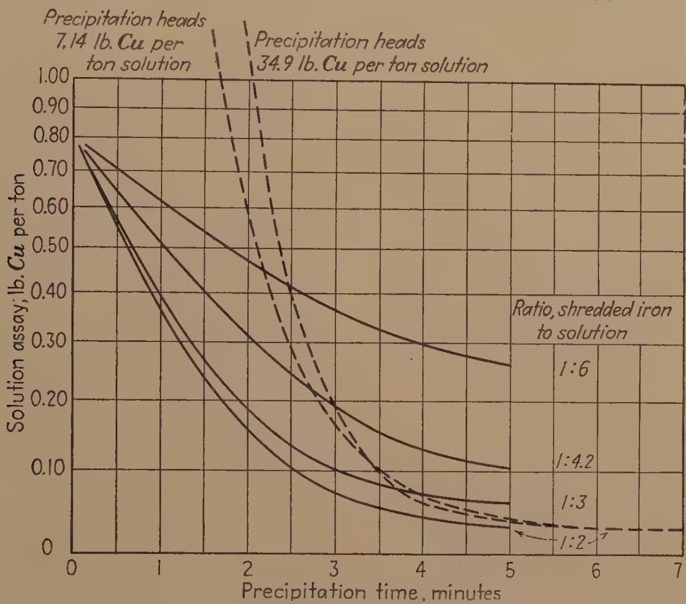

FIG. 3.—TIME-PRECIPITATION RELATIONS.
Solid lines indicate Ohio Copper pulps; broken lines, other pulps.

Iron consumption at the Ohio Copper plant is 2½ lb. per pound of copper precipitated—higher than usual in precipitation plants—because the iron is used to neutralize the free acidity from acid agitation. The iron is almost as cheap as lime for this purpose and does away with the necessity for accurate control at this point. The lime, if added, would have to be closely controlled to prevent precipitation of copper as the hydrate, which does not float.

The iron in the precipitator moves as a boiling mass near the bottom of the tank, giving excellent contact for precipitation. The intensity of action decreases near the top of the tank, until at the overflow the pulp is sufficiently quiet to drop out virtually all the iron.

The pulp from precipitation is fed into a 36-in. trommel, equipped with a screen having 0.5-in. openings. The purpose of this trommel is to remove the small amount of iron carried out of the last precipitation tank. The recovered iron is returned by hand. The pulp is sampled automatically ahead of flotation.

The flotation feed is split mechanically for roughing. Three Denver 8-cell 41½-in. flotation machines serve as roughers. Pentasol xanthate and amyl alcohol frother (0.04 and 0.15 per ton ore, respectively) are added to the splitter. The three rougher banks produce rougher concentrates and rougher tailings. The rougher concentrates are combined and cleaned in one 8-cell 41½-in. flotation machine. This machine produces final concentrates and cleaner tailings, which are added to the rougher tailings directly without re-treatment. To the combined rougher and cleaner tails is added the first trommel-screen oversize to give the final mill tailings.

There is no re-treatment, except one stage of cleaning, of any middling product. The entire rougher concentrate is cleaned and the cleaner tails discarded. When attempts have been made to re-treat any middling product, such as returning cleaner tails or scavenger concentrate to the head of the rougher circuit, less satisfactory over-all results have been obtained. Not only does the middling product fail to re-float, but it appears to have an adverse effect on new material.

It is necessary to maintain the highest possible grade of rougher concentrate to obtain good over-all results under these conditions. The cleaner tail consistently assays 0.4 to 0.6 per cent copper, regardless of the grade of feed to the cleaner bank; consequently, the more tonnage of this material, the greater will be the loss when discarding it.

At present, an 8 per cent rougher concentrate and 0.55 per cent cleaner tail gives a loss of only 4.6 per cent of the total copper (0.32 lb. per ton of ore) by discarding the cleaner tails.

The pH of the flotation circuit is kept about 4.7. Above pH 5.3, the flotation of

TABLE 2.—*Metallurgical Results, August–November, 1939*

Product	Solution			Solids			Total Solution + Solids				
	Tons	Cu per Ton, Lb.	Cu, Lb.	Tons	Per Cent Cu	Cu, Lb.	Percentage of Total Cu as Soluble	Percentage Cu Based on Solids	Cu, Lb.	Percentage of Total Based on Dry Tons	Percentage of Total Based on Cu Content
Mill Overall											
Concentrate......				1,340	25.37	679,900		25.37	679,900	1.17	72.38
Mill tails.........	305,000	0.110	33,600	113,260	0.100	225,800	12.95	0.115	259,400	98.87	27.62
Mill heads........	261,700	0.527	137,800	114,550	0.349	801,500	14.67	0.410	939,300	100.00	100.00
Leaching and Precipitation											
Mill heads.......	261,700	0.527	137,800	114,550	0.349	801,500	14.67	0 410	939,300	100.00	100.00
Trommel oversize.				4,500	0.220	19,800		0.220	19,800	3.93	2.11
Acid-leach feed[1]..	276,000	0.499	137,800	110,050	0.355	781,700	14.99	0.418	919,500	96.07	97.89
Precipitation feed[2]	276,000	0.736	203,200[6]	110,015	0.326	716,300	22.10	0.418	919,500	96.04	97.89
Flotation feed[3]...	276,000	0.125	34,600[6]	110,100	0.402	884,900	3.76	0.418	919,500	96.12	97.89
Flotation											
Concentrate......				1,340	25.37	679,900[8]		25.37	679,900	1.17	72.38
Flotation tails....	305,000[5]	0.110	33,600	108,760	0.095	206,000	14.02	0.110	239,600	94.94	25.51
Flotation feed....	276,000	0.125	34,600[7]	110,100	0.402	884,900[8]	3.76	0.418	919,500	96.11	97.89
Tailings											
Rougher tails[4]....	240,000	0.112	26,900	105,440	0.080	169,500	13.70	0.093	196,400	92.05	20.91
Cleaner tails[4].....	65,000	0.103	6,700	3,320	0.550	36,500	15.51	0.651	43,200	2.89	4 60
Trommel oversize.				4,500	0.220	19,800		0.220	19,800	3.93	2.11
Mill tails........	305,000	0.110	33,600	113,260	0.100	225,800	14.02	0.115	259,400	98.87	27.62

[1] Trommel undersize. Increase in water tonnage due to trommel spray water and Hydroseal gland water.
[2] Acid-leach tailings.
[3] Precipitation tailings.
[4] Distribution of rougher and cleaner tails approximate.
[5] Increase in water tonnage is from cleaner tailings.
[6] 82.93 per cent of soluble copper is precipitated.
[7] Some additional copper precipitation during flotation indicated.
[8] Indicated recovery of solids in flotation is 76.83 per cent.

cement copper is markedly retarded. On this ore, a pH below 5.5 is also beneficial for sulphide flotation, the low pH evidently assisting in maintaining sulphide surfaces free from oxide tarnish. Control of the flotation circuit is effected through:

1. Sand-slime ratio of feed. These terms although qualitative, have a definite meaning for the Ohio copper plant. Practically all portions of the dump can be classified as either sands or slimes; for good operation, one monitor sluices sands and the other slimes. If too high a ratio of slimes to sands is sluiced, the flotation froth becomes unmanageable and poor results are obtained.

2. Acidity. A free acidity of 0.5 to 1.0 lb. sulphuric acid per ton solution is maintained during acid agitation, the greater amount when the feed is higher in slimes.

3. Flotation reagents. Little manipulation of flotation reagents is necessary. The Minerec has proved an effective primary collector for both cement and sulphide copper in the presence of the soluble salts. The frother is regulated to give a light, quick-breaking froth, which is rapidly removed from the flotation machine.

Final flotation concentrates are thickened, after the addition of lime, in a 20-ft. thickener. The thickened pulp is filtered in a six-disk 4-ft. filter. Thickening and filtering characteristics are poor, as would be expected. The filter cake shipped averages about 30 per cent moisture.

TABLE 3.—*Distribution of Tailing Loss*

Item	Assay, Copper	Copper Loss	
		Percentage of Total in Mill Tails	Lb. per Ton Mill Tails
Rougher tails (solids), per cent	0.093	65.4	1.51
Cleaner tails (solids), per cent	0.550	14.0	0.32
Solution tails, lb. per ton	0.110	13.0	0.29
Trommel oversize, per cent	0.22	7.6	0.17
Mill tails, per cent	0.115	100.0	2.29

The initial operation of the plant was hindered by the corrosive action of the solutions, owing to their copper and acid content. Gradually rubber, copper and wood have been used to replace all other materials (except special alloys in a few instances) and satisfactory mechanical operation has now been established.

METALLURGICAL RESULTS

Table 2 shows metallurgical results for the period August through November,

TABLE 4.—*Operating Costs, August–November*, 1939

Item	Cost, Cents per Ton Ore Milled	
Sluicing		
Labor	4.51	
Labor insurance	0.25	
Power[1]	0.96	
Supplies	0.78	6.50
Milling		
Labor	3.75	
Labor insurance	0.21	
Power[1]	7.00	
Reagents	6.36	
Precipitant	3.18	
Supplies and miscellaneous charges	1.30	21.80
Mill Maintenance		
Labor	1.00	
Labor insurance	0.06	
Supplies	1.88	2.94
Tailings Disposal		
Labor	0.91	
Labor insurance	0.05	
Supplies	0.07	
Dike construction	0.85	1.88
Concentrate Haulage		1.77
Royalty		0.74
General Charges		
Supervision and assaying	1.55	1.55
Total operating costs		37.18 cents per ton milled.

[1] A saving of 20 per cent in the power cost will be made upon completion of arrangements to take delivery of power at 44,000 volts instead of 5,000 volts.

1939, treating 114,550 tons of ore. Over this period, a 72.4 per cent recovery of the total copper in the heads (assaying 0.41 per cent copper) was obtained in a 25.4 per cent copper concentrate. Final tailings assayed 0.115 per cent copper. Points of special interest in the metallurgical results are:

1. 15.7 per cent of the total copper in the mill heads was water-soluble and 22.1 per cent (1.8 lb. per ton of ore) was acid-and-water-soluble.

2. 82.9 per cent of the soluble copper was precipitated as cement copper in the precipitation operation.

3. Distribution of mill tailing losses is shown in Table 3.

The most important copper loss is that in the rougher tailings. Extensive testing is underway to improve results in the rougher circuit, but the outcome of this work is problematical. The copper in the rougher tailings is about 50 per cent oxide copper, but the acid concentration required to dissolve this portion is beyond economic limits. Improved metallurgical conditions for roughing might also reduce cleaner tailings losses. As stated, losses of soluble copper can be reduced, but mechanical changes to effect this reduction have not yet been made.

Operating Costs

Table 4 gives operating costs. Now that mechanical and metallurgical operations are established, it is expected that these costs will be reduced. Items not included are taxes and amortization.

Acknowledgment

Acknowledgment is made to Mr. Percy Kittle, President of the Ohio Copper Co., for permission to publish the foregoing material and for assistance in preparing the data and manuscript.

Results in the Duquesne Mill of the Callahan Zinc-Lead Company

By Joseph C. Kieffer,* Junior Member A.I.M.E.

(New York Meeting, February 1942)

The Duquesne property of the Callahan Zinc-Lead Co. is about 20 miles east of Nogales, in southern Arizona. One mine is near the mill, but most of the mill feed is hauled in by truck from a number of small mines.

The same sulphides are found in all the ores, in general, but the amounts vary considerably. All ores contain silver, copper, lead and zinc minerals but one mine ships ore high in silver and lead and low in copper and zinc; while another mine ships ore high in copper and zinc, and low in silver and lead. The common sulphides are galena, carrying most of the silver; chalcopyrite; sphalerite, carrying iron in solid solution; and pyrite, in a gangue consisting of limestone, quartz and garnet.

Mineralization is fairly coarse. At the present grind of about 66 per cent minus 200 mesh most of the sulphides are liberated. A small amount of chalcopyrite is locked up with the sphalerite and cannot be liberated at an economic grind, and the copper carried into the zinc concentrate with the sphalerite accounts for the comparatively poor recovery of copper in the copper-lead concentrate.

Production of Two Concentrates

The milling problem, which was complicated by the extreme variation in the grade of the feed, was first attacked with the idea of producing a silver-copper-lead concentrate and a zinc concentrate, both with satisfactory grades and recoveries. By February 1941, it was felt that these objectives had been attained, when the grades of the various products and the recoveries were as shown in Table 1.

Table 1.—*Grades of Products and Recoveries*

Mill Assays

Products	Ag, Oz.	Cu, Per Cent	Pb, Per Cent	Zn, Per Cent
Heads (120 tons daily)	4.86	0.95	2.9	11.5
Cu-Pb concentrate	71.3	12.1	44.1	6.6
Zn concentrate	1.3	0.8	0.6	55.7
Tailings	0.19	0.07	0.08	0.69

Mill Recoveries

Products	Ag, Per Cent	Cu, Per Cent	Pb, Per Cent	Zn, Per Cent
Cu-Pb concentrate	91.9	79.2	94.1	3.6
Zn concentrate	5.1	15.6	4.0	92.0
Tailings	3.0	5.2	1.9	4.4

A rough outline of the flowsheet and reagents used is as follows: Minus ¾-in. feed to a 7-ft. by 36-in. Hardinge ball mill working in closed circuit with a 36-in. Wemco classifier. The classifier overflow passes to the second cell of an 8-cell bank of No. 18 Special Denver Sub-A flotation machines (volume, 22.5 cu. ft. per cell). A rougher concentrate is removed from the second to the fifth cells inclusive, and returned to the first cell, which is the cleaner. A scavenger concentrate is removed from the sixth, seventh, and eighth cells, and returned to the third cell. Consump-

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tions in pounds per ton of ore of reagents used in the copper-lead section are as follows: To the ball mill, 0.35 zinc sulphate, 0.08 sodium cyanide, and 0.10 isopropyl xanthate; to the second cell, 0.09 DuPont frother B-48; to the third cell, 0.04 isopropyl xanthate; to the fourth cell, 0.05 DuPont frother B-48; and to the sixth cell, 0.04 isopropyl xanthate. The pH in this circuit is about 7.8.

The copper-lead tailings are pumped to a 6 by 6-ft. Wemco conditioner, the overflow of which passes to an 8-cell No. 18 Special Denver flotation machine. A rougher concentrate is taken from the first five cells and is treated in a three-cell cleaner of the type that is used for roughing. A scavenger concentrate is taken off the last three cells and returned to the third rougher cell. The cleaner tailings are fed into the second rougher cell. Consumptions in pounds per ton of ore of reagents used in the zinc section are as follows: To the conditioner, 2.5 lime, 1.0 copper sulphate, 0.08 sodium cyanide, 0.11 amyl xanthate, and 0.06 pine oil; and to the fourth rougher cell, 0.04 amyl xanthate. The pH in the zinc circuit is around 11.2.

The percentages of solids carried at different points throughout the circuit are as follows: ball-mill discharge, 77; classifier overflow, 35; copper-lead tailings, 26; and zinc tailings, 19.

Separation of Copper-lead Concentrate

When these results had been obtained, the next problem to be solved was the separation of the copper-lead concentrate into a copper concentrate and a lead concentrate. Very little has been published on such work, so the whole problem had to be worked out. A great deal of laboratory work was done, with the idea of finding out as much as possible about reagent combinations and control for the separation.

Sodium dichromate was the first reagent tried, the intention being to depress the galena and allow the chalcopyrite to float, an operation with which the author had had some previous experience. Separations were attained but the results were erratic.

The next reagent tried was sodium cyanide, the plan being to float the galena and depress the chalcopyrite. Results were better and could be consistently repeated. When this method was sufficiently perfected, it was put into practice in the mill.

Five No. 18 Special Denver Sub-A flotation cells handle the separation. The first cell is the cleaner, producing the finished lead concentrate; the second cell, which has the circulating gate opened so that no froth will be discharged at the overflow lip, is the conditioner; and the last three cells pull a rougher concentrate which is sent to the cleaner. The tailing of this bank is the copper concentrate. The copper-lead bulk concentrate is fed into the conditioner cell, as is also the cleaner tailing. A standard 6 by 6-ft. conditioner was first employed in the separation, giving a conditioning time of about 4 hr. This proved too long, resulting in excessive consumption of cyanide ion and giving the galena a chance to oxidize slightly. The 20-min. conditioning time secured by using a flotation cell of 22.5 cu. ft. capacity as a conditioner has proved to be ample.

Sodium cyanide is the only reagent used in the separation, about 0.75 lb. per ton of ore being fed into the conditioner cell and 0.25 lb. into the cleaner cell. This is the equivalent of about 11 lb. of sodium cyanide per ton of copper-lead concentrate to the conditioner, and 4 lb. of sodium cyanide per ton of copper-lead concentrate to the cleaner. Typical titrations for free NaCN, titrating against silver nitrate, of the filtrate of grab samples taken in various points of the circuit show the following: cleaner cell, 0.015 per cent; conditioner cell, 0.022; first rougher cell, 0.015; second rougher cell, 0.015; and third rougher cell, 0.015. Density in the conditioner is held around 1.130. Representative assays for

the separation (July 1941) are shown in Table 2; also recoveries for the same period and recoveries attained by the separation on its feed—i.e., the copper-lead concentrate.

TABLE 2.—*Separation of Copper-lead Concentrate*

ASSAYS OF BULK CONCENTRATE AND SEPARATION PRODUCTS

Product	Ag, Oz.	Cu, Per Cent	Pb, Per Cent	Zn, Per Cent
Cu-Pb concentrate....	77.82	17.63	37.57	4.22
Pb concentrate.......	130.29	2.38	70.96	3.17
Cu concentrate.......	9.90	28.01	3.40	5.54

RECOVERIES IN BULK CONCENTRATE AND SEPARATION PRODUCTS BASED ON HEADS

Product	Ag, Per Cent	Cu, Per Cent	Pb, Per Cent	Zn, Per Cent
Cu-Pb concentrate....	93.0	86.6	94.1	3.7
Pb concentrate.......	85.4	5.9	89.1	1.2
Cu concentrate.......	7.6	80.7	5.0	2.5

RECOVERIES IN SEPARATION PRODUCTS BASED ON BULK CONCENTRATE

Pb concentrate.......	91.8	6.8	94.7	32.4
Cu concentrate.......	8.2	93.2	5.3	67.6

About 0.08 lb. of copperas ($FeSO_4 \cdot 7H_2O$) per ton of ore, corresponding to around 5 lb. per ton of copper concentrate, is added to the copper-concentrate thickener to facilitate settling and to kill any free cyanide present.

The two most important points in the control of the separation are the pH and the CN^- ion concentration. The pH is maintained in the range 9.7 to 10.2. Keeping in mind the dissociation constant for hydrocyanic acid, it can be shown that the lower the pH value of an NaCN solution of given strength, the lower the concentration of CN^- ions. This principle comes into play sufficiently at a pH of under 9.7 to prevent the proper depression of chalcopyrite, allowing too much copper to appear in the lead concentrate. On the other hand, if the pH climbs above 10.2, the galena is depressed sufficiently to throw too much lead into the copper concentrate. Fortunately, sodium cyanide furnishes both hydroxyl ions for the pH control and cyanide ions for the depression of chalcopyrite in, under normal conditions, about the proper proportions. On only one occasion, when oxidized ore appeared in the feed, has difficulty been encountered in maintaining proper conditions in the separation. On this occasion there seemed to be copper minerals present in the copper-lead bulk, which, while not soluble enough to activate the sphalerite in the copper-lead circuit, were more subject to attack by cyanide solution than pure chalcopyrite. As a consequence, results in the copper-lead circuit were satisfactory, while in the separation, even after the amount of sodium cyanide fed had been raised high enough to bring the pH over 10.2, with consequent depression of galena into the copper concentrate, the CN^- ion concentration was too low to prevent an excessive amount of copper from appearing in the lead concentrate. This condition existed for a period of several days, and several remedies were tried, including pH control with sulphuric acid. By far the most successful thing tried, and the only one that gave satisfactory results, was the increase of the amount of cyanide fed into the ball mill to several times the normal amount used and control of the separation with sodium cyanide alone, as is normally done.

The explanation for this seems to be that in this way all the copper easily soluble in cyanide was removed in the copper-lead circuit, so that actual consumption of CN^- ion in the separation was no more than usual. Careful panning of products in the copper-lead circuit was carried on during this period, to be sure that as much cyanide as possible was being added to the ball mill without depressing the copper in the bulk circuit.

Since of all the common copper sulphide minerals chalcopyrite is attacked least by cyanide, it is not likely that any other means of control will be required than that now employed. However, with other copper minerals, the addition of buffer salts in the pH range of 9.7 to 10.2 may be found helpful when sodium cyanide is employed to separate copper sulphides from galena.

Crude cyanide (Aero brand), which raises the pH more than sodium cyanide for equal contents of cyanide ion, gave good results in test work but was not tried in the mill. It is believed that its use may in some cases effect a cost saving in the separation of chalcopyrite-galena concentrates, especially where the ratio of the former to the latter is low. Its use would be limited, of course, to very clean sulphide ores.

Milling Methods at the Lead-zinc Concentrator of Compañia Minera de Peñoles, S. A., at Avalos, Zacatecas, Mexico

By Irving M. Symonds,* Member A.I.M.E.

(New York Meeting, February 1942)

Compañia Minera de Peñoles, S. A., at its Avalos unit operates a lead-zinc concentrator having a capacity of about 600 short tons per day. Lead and zinc concentrates are made by flotation methods. The concentrator is at Terminal de Providencia, in the state of Zacatecas, some 70 miles south of Saltillo, Coahuila. The Peñoles-Avalos railroad operating between the plant and Estacion Avalos, a distance of 6 miles, makes connections with the Coahuila and Zacatecas railroad, which in turn connects with the Nacionales de Mexico at Saltillo. The three railroads are each of different gauge, requiring the transfer of all supplies and concentrates at each junction.

The mill operates on company ores, on ores from leased mines, and as a custom mill. The custom business is very small. The lead concentrates are shipped to company smelters at Monterrey, Nuevo León and Torreón, Coahuila. Since the European war, all zinc concentrates have been shipped to the United States.

Ore Treated

The major source of ore supply is a group of four mines owned or leased by the company, some three miles south of the mill in the mountains, from which the ore is delivered to the mill by a gravity aerial tramway. Custom ore comes in by rail.

The ores occur in chimneys. The upper levels contained oxidized siliceous ores of lead, copper, silver and gold, with varying amounts of zinc. These ores were direct-smelted. In the first two years of operation a good deal of the tailing loss was caused by the oxidized content of the ores milled. Much of the ore in the transition zones between the oxides and sulphides was not mined.

The ores now being milled are clean, hard sulphides in a limestone gangue, with only occasional contamination of oxides. The lead and zinc sulphides are of coarse structure and are freed even above flotation sizes. The galena is usually "steel" colored, easily and quickly floated. When there is any oxidation, it is usually on the coarsest crystals. The zinc minerals vary a good deal in color; a resin sphalerite predominates but there are varying shades of brown marmatites. On an average, the zinc sulphides contain 64 per cent zinc. The only common copper mineral is chalcopyrite. Little is known of the occurrence of gold and silver. Waste minerals are mainly limestone and pyrite, both of which are barren. The ores vary from 20 to 35 per cent pyrite.

Mill heads for the year 1940 and the first eight months of 1941 are given in Table 1.

Table 1.—*Mill Heads*

Year	Ounces per Ton		Per Cent	
	Gold	Silver	Lead	Zinc
1940	0.016	8.23	9.4	16.3
1941 (8 months)	0.013	7.15	7.6	17.3

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History

The concentrator was built in 1934 and 1935. Operation started in January 1936 and has been continuous to date. Almost all the equipment had been used by the company at another location.

Losses were high in the first two years of operation, owing to an untrained operating crew, oxidation of the ore, mechanical difficulties, and a flowsheet not well suited for the ores. Time, of course, gave the operating force experience and improved the quality of the mill feed, as the ores near the transition zone were exhausted. The mechanical difficulties were gradually overcome. Many changes have been made in the flowsheet, both minor and major. Little new equipment has been added to the concentrator, but some modifications of old equipment have greatly improved results. For the past three years results have been quite satisfactory.

Table 2 summarizes the results obtained during the life of the concentrator, giving concentrate grades, metal recoveries and tonnage* milled.

Main improvements in metallurgy and tonnage have come from finer crushing by better utilization of the crushers; returning to primary rod milling after experimenting with primary ball milling; better classification by local improvements on old-type Dorr classifiers; introduction of lead flotation between two stages of grinding; replacement of Forrester flotation machines used as scavengers in both the lead and zinc circuits by mechanical machines; modifications of the Minerals Separation subaeration machine in use; introduction of tabling on lead flotation middlings; and by minor but important changes in control of flotation reagents.

Crushing

Primary crushing to 3 in. is done at the mine in a jaw crusher. At this point up to 10 per cent of the run-of-mine ore, mostly barren limestone, is removed as waste by hand sorting.

The ore is delivered to the mill by a gravity aerial tramway. The ores are weighed on a tramway scale at the mill terminal of the cableway, at which point the moisture samples are taken. The storage ahead of the mill crusher consists of seven wooden ore bins having a total capacity of 500 short tons. It is necessary to keep the ores from each mine separate until after fine crushing, as weights, moisture samples, and assay samples are taken to determine royalty payments on ores from leased mines.

Fig. 1 shows the present flowsheet of the crushing plant. Two stages of crushing are

TABLE 2.—*Average Tonnage[a] and Metallurgical Results during Life of Concentrator*

Year	Tons Milled	Tons per Day[b]	Lead in Lead Concentrate, Per Cent	Recovery in Lead Concentrate, Per Cent: Silver	Recovery in Lead Concentrate, Per Cent: Lead	Zinc in Zinc Concentrate, Per Cent	Recovery of Zinc in Zinc Concentrate, Per Cent
1936	63,040	230	53.8	78.7	82.8	54.1	73.2
1937	87,950	252	60.7	82.2	86.8	54.1	83.5
1938	85,291	239	63.0	87.3	90.9	59.3	86.5
1939	87,860	290	64.8	91.5	96.0	59.5	88.4
1940	109,351	407	67.5	92.3	97.9	59.4	90.3
1941 (8 mo.)	95,119	476	69.5	90.7	97.3	60.9	93.0
Aug. 1941	13,477	518	70.8	90.2	97.1	61.6	93.2

[a] All tonnage figures in this paper are for the ton of 2000 pounds.
[b] The tons per day is not the tonnage milled for 24 hr. of operating time, as it does not take into consideration lost time. At present the concentrator is not run on Sundays and most repair work is done on Mondays. The actual tonnage per 24 hr. of operating time is now 40 to 50 tons above the figure given. For August 1941 it was 566 tons. The record rate to date is 623 tons.

* All tonnage figures used in this paper are for the ton of 2000 pounds.

practiced: a gyratory with curved concaves and head breaking to ⅞ in. and rolls which further reduce the product until it all passes a ⅜-in. screen. The feed to each crusher is screened ahead of the machine. The assay sample is taken from the finished product by an automatic straight-line cutter at regular time intervals, before the fine ore is conveyed to the mill storage bins.

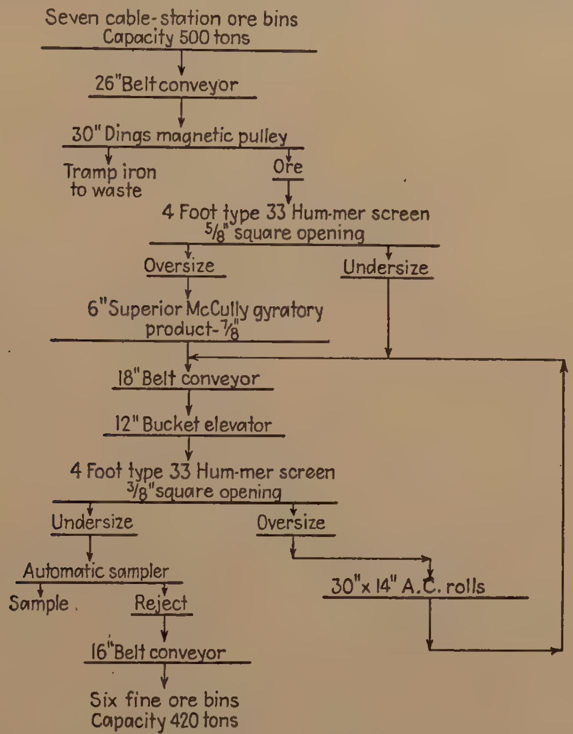

FIG. 1.—FLOWSHEET OF CRUSHING PLANT.

The crusher usually can handle 200 to 250 tons per shift, depending mainly on the moisture content of the ore. Three operating shifts are maintained, which gives time to keep the plant clean and to do most repair work on day shift without interfering with mill operations.

Several changes have been made in the crushing plant since operations started, Originally a finished product that passed an opening 1 in. square was made, but gradually it has been reduced until now it passes a ⅜-in. screen. The reduction of the size of the finished product has greatly reduced the hourly capacity of the crushing plant and required more maintenance, especially on the rolls. However, what is more important, it breaks up the tough blocky pieces of ore; which, if done in wet grinding, would greatly reduce the capacity of the grinding units and tend to overgrind the fragile galena. Even finer crushing would be done if it were practicable with the available equipment.

GRINDING AND CLASSIFYING

The six concentrator ore bins have a capacity of 420 short tons, but only about 325 tons is available without shoveling. The ores are blended as far as the limited storage capacity allows. As often about 600 tons per day is milled, and as ores from the

various mines are received irregularly, blending is more of a hope than a fact. Head assays on lots from different mines often vary 100 per cent in lead content and 50 per cent in zinc. However, ore is always fed from all six ore bins to be conveyed to the rod mill.

The flowsheet of the grinding section is shown in Fig. 2. Two stages of grinding are now practiced. The primary rod mill is in closed circuit with its Dorr classifier by means of a Wilfley pump, and the secondary overflow ball mill with its classifier by similar means. Most of the lead and silver minerals are floated from the primary classifier overflow before it enters the secondary classifier. The reasons for this will be discussed in connection with the lead concentration circuit.

Undoubtedly the greatest improvement that has been made in the grinding circuit since the mill began work has been in classification. There was available at the start of operations a 6-ft. wide model D duplex Dorr classifier, which later was supplemented by a model C duplex Dorr 4 ft. 6 in. wide. The model C was placed in closed circuit with the rod mill and the model D with the ball mill. The model C was so light that it would stall if the rakes were slightly overloaded. Its practical raking capacity was only 7 or 8 tons of sand per hour. The model D would rake from 12 to 15 tons per hour. It was obvious that more circulating load would be beneficial, but at that time new equipment seemed out of the question.

Therefore it was decided to try to draw off some sand from the lower end of the pond, through pipes inserted through and welded to the sheet-iron box. All the experimental work was done on the primary classifier. As the classifiers were above the mills, the sands could be returned to the

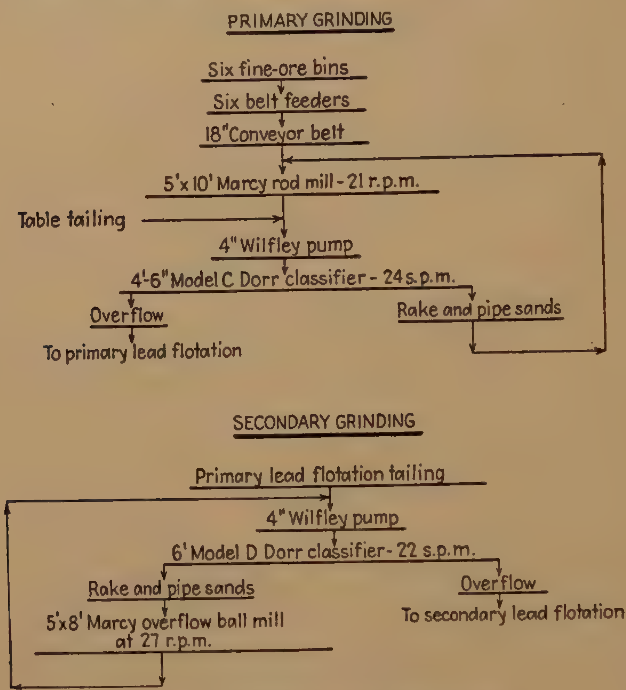

FIG. 2.—FLOWSHEET OF GRINDING UNITS.

mill scoop boxes by gravity. At first one ⅜-in. pipe at the center line of each rake was inserted into the box below the overflow lip, so that the inside end of the pipe

After the method was properly developed on the primary classifier, it was successfully installed on the secondary classifier. However, as there is no dry feed to keep the

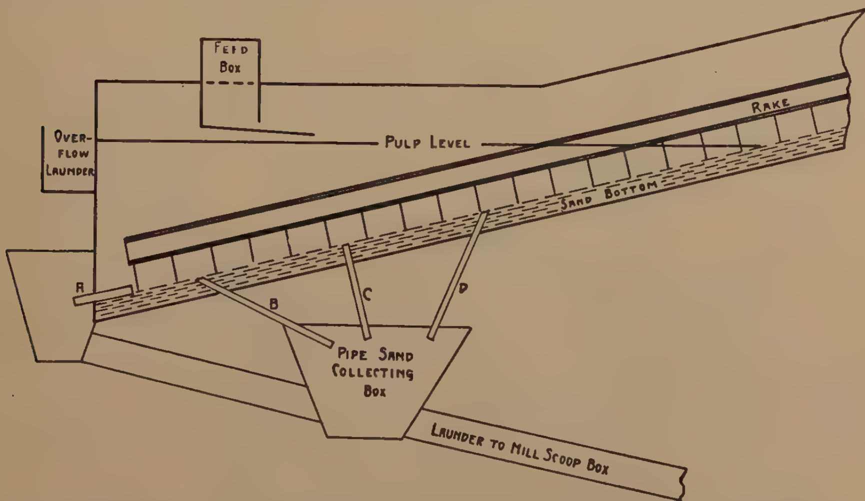

FIG. 3.—SECTION THROUGH PRIMARY CLASSIFIER SHOWING SAND-DISCHARGE PIPES.
Pipes are placed in rows across width of classifier. On row A usually three pipes are used, one in each corner and one in the center. Rows B, C, and D have four to a row and all pipes are used. On the secondary classifier only rows B and C are used, six pipes to a row.

just missed the rake at its lowest and hindmost position, as shown by pipe A in Fig. 3. Heavy sands were drawn off but the pipes choked up frequently. The next step was larger and more pipes. Then it was decided to go under the pond and draw off sand at evenly spaced intervals through pipes from under the rakes, as shown by pipes B, C, and D in Fig. 3. It was believed that these pipes would choke less and the sands would be denser. This proved to be true. In the primary classifier 14 to 16 one-inch pipes are now used, most of them under the pond. The pipe sands and the rake sands have identical screen analyses. The amount of sands drawn through the pipes is seven to eight times the amount dragged by the rakes. In operation the rakes are kept well loaded and the pipe sands stay about 80 per cent solids. No fresh water is added to the dry feed entering the rod mill, and a satisfactory mill dilution is thus obtained.

dilution down in the secondary ball mill only two to three times as much sand can be drawn through the pipes as the rakes will handle.

The improved classification has greatly increased the capacity of the mill, undoubtedly over 200 tons per day. It has also given better metallurgical results. It is very possible that modern classifiers of high sand-raking capacity would duplicate the results. However, the mill has considerably exceeded in tonnage and metallurgy the estimates that a representative of one well-known classifier manufacturer made for using his company's new-type equipment. The classifiers now "kick" over coarse, free limestone while retaining the sulphides until they are fine enough to float. The larger circulating load reduces overgrinding of galena and gets it into the classifier overflow at a coarser average size.

Some data on the grinding mills as now operated are given here. The rod mill is a 5 by 10-ft. Marcy with the discharge opening closed up to 12 in., scoop fed, and running at 21 r.p.m. Carbon-steel rods $2\frac{1}{2}$ in. high are used to maintain the load; they are added daily. Manganese-steel liners last from 550 to 600 operating days as breast and feed-end liners, and from 200 to 300 days as discharge-end liners. Liner consumption is 0.11 lb. per ton at a cost of $0.015 U. S. currency per ton.

The secondary grinder is a 5 by 8-ft. overflow ball mill, scoop fed, 12-in. discharge opening, running at 27 r.p.m. The replacement balls are 2-in. cast-iron balls, and the mill is kept full. All liners are of manganese steel; they have a life from 350 to 600 days, with an average consumption of 0.10 lb. per ton at a cost of $0.014 per ton. Both mills are driven by identical 100-hp. motors through reduction gears. Consumption of total grinding media is 1.56 lb. per ton at a cost of $0.04 per ton of ore. All liners and grinding media are of Mexican manufacture.

Early in the life of the concentrator, an overflow ball mill was tried as a primary grinder in place of the rod mill. The ball mill would not break up the coarse limestone as well as the rod mill; in fact, it appeared to be more of a wearing down process than grinding. By limiting the feed to capacity of the mills to break the limestone, much of the galena would be overground. A grate mill of proper diameter should give good results, but there is no doubt that the 5-ft. rod mill will handle more tonnage and grind the Avalos ores so that better metallurgy can be obtained than will an overflow ball mill of the same diameter.

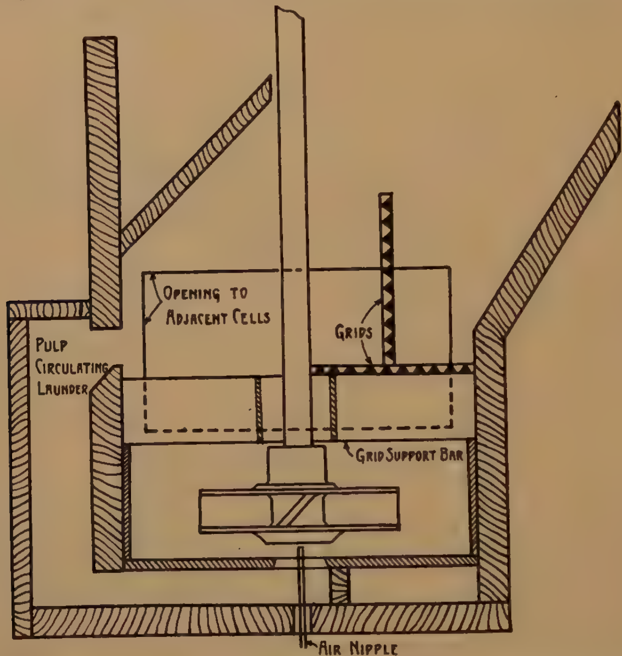

Fig. 4.—Modifications to Minerals Separation flotation machine.

Concentration

The Flotation Machine

The original flotation machines used were 18-in. subaeration Minerals Separation type converted to Hebbards. The

Hebbard is a Minerals Separation cell with openings for the passage of pulp from cell to cell above the side liners, the opening usually being from 12 to 14 in. square. It side of the cell, and returned under the bottom liner to the center of the impeller of the same cell. The impeller, acting as a pump, has the capacity to circulate several

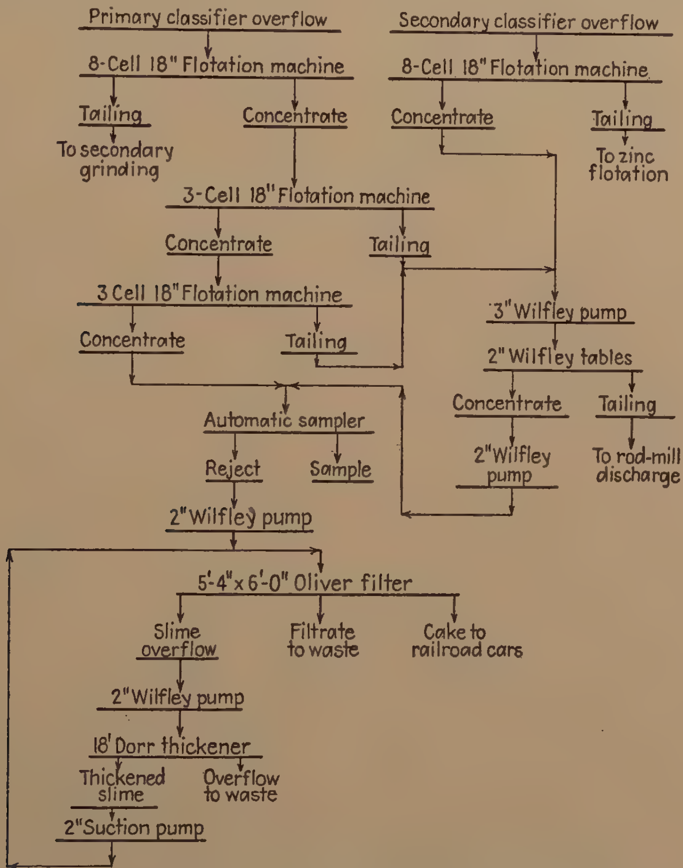

FIG. 5.—FLOWSHEET OF LEAD CONCENTRATION AND FILTRATION.

uses no overflow weir except on the last cells, to maintain the pulp level. Both the Hebbard and the standard Minerals Separation cells choked up badly on the heavy ore, especially as coarser grinding was adopted.

To overcome the choking and to make a satisfactory cell that works freely, a circulating launder was put on the back of each cell, as illustrated in Fig. 4. The pulp is taken out of an opening in the back of each cell, about 16 in. above the bottom liner, into an enclosed launder on the out- times the tonnage of the original feed, thus keeping the heavy pulp well agitated and in suspension. Instead of four grids, as is customary, only two are used: one horizontal on the froth-discharge side resting on the usual grid-support bars and the other vertical resting on the horizontal grid at right angles to the froth flow. The grids, as shown in Fig. 4, allow the pulp to be better agitated than the old method, while still giving a quiet froth surface. Froth paddles are unnecessary. The opening between cells has been varied and some

experiments are still going on, but a large opening 14 in. high by 30 in. wide has been found quite satisfactory. The pulp is discharged from the last cells by weirs about one half the width of the standard Minerals Separation weirs. The machine was developed step by step and is now believed to be well suited for the heavy Avalos ores and to be capable of giving excellent metallurgy at a reasonable operating cost.

In the original flowsheet there were two 30-ft. Forrester flotation machines, intended to be used as scavengers in both the lead and zinc circuits. These machines choked so badly and gave such poor results that each one was finally replaced by eight cells of the type of machine described above.

Lead Concentration Flowsheet

In Fig. 5 is shown the lead concentration and filtration flowsheet. The primary classifier overflow is floated in eight cells, the froth from which is cleaned and recleaned. Each cleaning step is done in three cells.

The secondary classifier overflow is floated in eight cells, which makes a tailing that is the zinc flotation head. The concentrate from the secondary lead flotation joins the two cleaner tailings and is pumped to two Wilfley tables. The tables make a finished lead cut, while the table tailings enter the rod-mill discharge pump to go to the primary classifier as diluent.

TABLE 3.—*Reagent Consumption for Period January to August* 1941, *Inclusive*

Reagent	Pounds per Ton	Cost per Ton
Lead flotation:		
Cresylic acid	0.11	$0.010
Hydrated lime	0.32	0.002
Zinc sulphate	0.91	0.032
Sodium cyanide	0.07	0.011
Secondary butyl xanthate	0.04	0.011
Total lead flotation	1.45	0.066
Zinc flotation:		
Sodium cyanide	0.03	0.005
Hydrated lime	1.00	0.007
Copper sulphate	0.56	0.028
Sodium ethyl xanthate	0.12	0.018
Total zinc flotation	1.71	0.058
Grand total	3.16	$0.124

The flotation reagents used are shown in Table 3, and there is little about them that is not standard practice. Cresylic acid is added to the rod mill. Lime, zinc sulphate, sodium cyanide, and secondary butyl xanthate are just dumped into the primary classifier overflow. Small amounts of cresylic acid and secondary butyl xanthate are added to the secondary classifier overflow. Points of addition of various reagents have been changed from time to time, but present practice appears to be the simplest and most effective. The primary lead tailing is maintained from pH 7.7 to 8.0.

The unusual practice of floating most of the lead and silver minerals at a coarse grind from the primary classifier overflow, and recovering the remainder from a finer grind, deserves some discussion of the reasons for doing so. Three main periods in the development of the present flowsheet should be considered:

1. Two stages of grinding with lead flotation after grinding was completed.

2. The same two stages of grinding but with lead flotation after each stage.

3. The same as the second period but after classifier improvements had been made.

Table 4 shows typical screen analyses of the lead concentrates from each of these three periods. The minus 200-mesh portion of the lead concentrate has been reduced from 89.4 per cent in the first period to 77.1 per cent in the second, and finally to 64.2 per cent in the third. The second period shows a substantial improvement in lead recovery and some tonnage increase over the first period; while the third period brings about a general improvement in metallurgy with a large tonnage increase.

There can be no doubt that much of the metallurgical improvement has come from the elimination of unnecessary grinding

and more selective grinding. Any unnecessary grinding is bound to cause some flotation loss. The primary flotation removes lead and silver minerals of floatable sizes that otherwise would get into the secondary classifier sands and be further reduced in the ball mill. The improved classification "kicks out" the light, barren limestone from the grinding circuit at a much coarser size than formerly. The work thus eliminated is applied to sulphide minerals, allowing a great increase in capacity. The heavy circulating loads reduce unnecessary grinding on sulphide minerals to a minimum. It is interesting to note that the same number of flotation cells can handle a much larger tonnage of lead than on the older flowsheets, although it was necessary to develop the present flotation cell to handle the coarser feed.

The primary classifier overflow varies a great deal in screen analysis. A high-limestone ore with low pyrite makes a much coarser overflow than low limestone with high pyrite. Samples of the primary overflow have shown 25 to 40 per cent minus 200-mesh, with the average about 35 per cent. Between 80 and 90 per cent of the galena is minus 100-mesh, while virtually all samples show some limestone coarser than 14-mesh and even traces on 10-mesh. The tailing from primary flotation averages 1.75 oz. of silver and 1.0 per cent lead. From 50 to 75 per cent of the lead in the primary tailing is in the sizes too coarse to float well.

The secondary classifier overflow contains from 40 to 45 per cent minus 200-mesh. The lead load in secondary flotation is usually very light, but because of wide variation in lead content of the original feed, the underloaded secondary flotation serves as a protection to catch any lead that may be forced into the primary tailing by a sudden increase in head assay.

Tables were installed in the mill to make a lead cut on primary-mill discharge, but only a very small amount of lead could be taken off if a good concentrate grade were to be maintained, and the lead removed seemed to be that which was most easily floated. However, it was found that

TABLE 4.—*Screen Analyses of Lead Concentrates Made during Different Periods of Mill Operations*

Mesh	Period No. 1 Flotation after Two Stages of Grinding Completed but before Classifier Improved		Period No. 2 Flotation after Each Stage of Two-stage Grinding but before Classifier Improved		Period No. 3 Flotation after Each Stage of Two-stage Grinding with Improved Classifier	
	Per Cent Weight		Per Cent Weight		Per Cent Weight	
	Individual	Cumulative	Individual	Cumulative	Individual	Cumulative
+ 65			0.8	0.8	2.4	2.4
− 65 + 100			3.3	4.1	8.4	10.8
−100 + 150	2.4	2.4	7.9	12.0	12.3	23.1
−150 + 200	8.2	10.6	10.9	22.9	12.7	35.8
−200 + 325	89.4[a]		12.7	35.6	14.5	50.3
−325			64.4		49.7	
Total	100.0		100.0		100.0	
Lead recovery, per cent	91.5		95.8		97.1	
Lead in lead concentrate, per cent	63.0		61.5		70.8[b]	
Tons ore milled per day	244		287		518	

[a] Sample was not screened on 325 mesh. All samples wet and dry screened on finest screen.
[b] During this period recleaning of the lead concentrate was started. Before recleaning the grade was about 67 per cent lead.

tabling on the cleaner tailings and the secondary lead concentrate gave excellent results. It removed some lead that was slightly tarnished and therefore hard to secondary classifier overflow, which usually is over 40 per cent solids.

Cleaning of the lead concentrate was started in 1937. It was not entirely success-

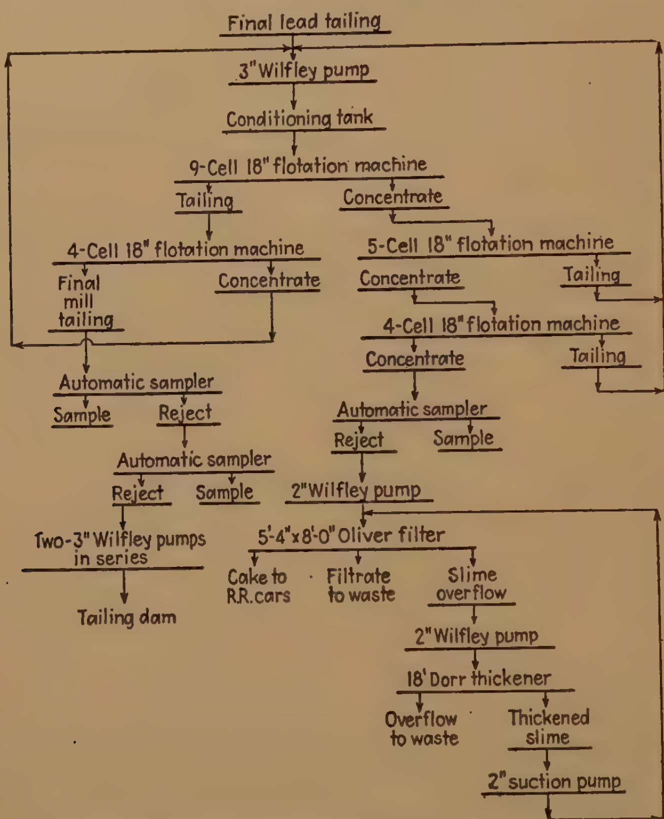

FIG. 6.—FLOWSHEET OF ZINC FLOTATION AND FILTRATION.

clean up to a good grade. Lead and silver extraction increased about one per cent when the practice was introduced. The concentrate is lower in zinc than the finished flotation concentrate. By using the table tailing as diluent in the primary classifier, the dilution caused by cleaning and launder sprays in the lead circuit is avoided. This is very important where such a large percentage of the original feed is removed as concentrate. As very little froth is removed in the secondary lead-flotation machine, the pulp enters the zinc circuit at almost the same dilution as the ful until tabling was done on the cleaner tailing several months later. Decreased lead content of the ore allowed cells to be used for recleaning in the early part of 1941. The recleaner tailing was also passed over the tables.

Analyses of lead concentrate are given in Table 7.

Zinc Flotation Circuit

The present zinc flotation and filtration circuit is shown in Fig. 6. The final lead tailing is conditioned approximately one hour. The conditioner discharge is floated

in a nine-cell rougher machine. The froth from the rougher is cleaned in five cells and recleaned in four cells. The tailing from the rougher enters a four-cell scavenger machine, which makes the final mill tailing. The froth from the scavenger and both cleaner tailings are returned to the conditioner.

TABLE 5.—*Typical Screen Analysis of Zinc Concentrate*

Mesh	Per Cent Weight	
	Individual	Cumulative
+ 65	8.3	8.3
− 65 + 100	17.3	25.6
−100 + 150	16.0	41.6
−150 + 200	13.9	55.5
−200 + 325	14.0	69.5
−325	30.5	
Total	100.0	

The flotation reagents used are shown in Table 3; they are about standard for zinc sulphide flotation. Lime and a small amount of cyanide are added to the lead tailing before it enters the conditioner. The conditioner discharge is held about pH 9.3. Copper sulphate and sodium ethyl xanthate are added to the conditioner discharge. Small amounts of copper sulphate and ethyl xanthate are added to the feed of the scavenger machine. It is often necessary to use traces of xanthate in the feed to either or both cleaners.

Both cleaning and recleaning of the zinc concentrate have been introduced since the mill started. Cleaning was first practiced in the latter part of 1937, while recleaning was started early in 1941, when four additional cells were added to the circuit. The grade of concentrate and the recovery of zinc have justified the changes that have been made.

Table 5 gives a typical screen analysis of the zinc concentrate from present practice.

Final tailing analyses from present practice show that over 90 per cent of the lead and approximately 25 per cent of the zinc losses are as oxidized minerals. Table 6 gives typical analyses and assays of the final tailings. Table 7 shows analyses of flotation products and mill recoveries.

Under present conditions of tonnage and ore, the zinc circuit is working near capacity, while the lead circuit can well handle half as much again of lead content as it now has on an average.

TABLE 6.—*Screen Analyses and Assays on Final Mill Tailing from Present Practice*

Mesh	Per Cent Weight		Assays			Distribution, Per Cent					
						Silver		Lead		Zinc	
	Individual	Cumulative	Silver, Oz.	Lead, Per Cent	Zinc, Per Cent	Individual	Cumulative	Individual	Cumulative	Individual	Cumulative
+ 48	7.0	7.0	0.35	0.05	1.8	3.9	3.9	3.5	3.5	12.4	12.4
− 48 + 65	13.6	20.6	0.67	0.08	1.3	14.3	18.2	9.7	13.2	17.4	29.8
− 65 + 100	16.4	37.0	0.73	0.10	0.9	18.8	37.0	14.2	27.4	14.6	44.4
−100 + 150	12.6	49.6	0.67	0.07	0.6	13.2	50.2	8.0	35.4	7.5	51.9
−150 + 200	11.2	60.8	0.58	0.06	0.6	10.2	60.4	6.2	41.6	6.6	58.5
−200 + 325	12.4	73.2	0.58	0.06	0.6	11.3	71.7	6.2	47.8	7.2	65.7
−325	26.8		0.67	0.22	1.3	28.3		52.2		34.3	
Total	100.[illegible]		0.64	0.11	1.02	100.0		100.0		100.0	

DISPOSAL OF CONCENTRATES

Originally the concentrates were thickened in 18-ft. Dorr thickeners, one each for the lead and zinc, and filtered in Oliver filters, that for the lead being 5 ft. 4 in. by 6 ft. and the one for the zinc 5 ft. 4 in. by 8 ft. The system worked well on light

tonnage, but when the production of zinc concentrates was more than 75 tons per day the thickener was overloaded. To avoid the difficulty, the concentrate was pumped directly to the filter. Most of the coarse concentrate was removed immediately in the cake. As the filter could then not handle all the water put into it, an overflow pipe was placed about halfway up the side of the filter tank, which discharges the excess water and some slime zinc concentrate. The overflow is pumped to the thickener and the thickened slimes returned to the filter by a suction pump. The system has handled more than 2500 lb. of concentrate per 24 hr. per square foot of filter canvas. It proved so trouble free on the zinc concentrate circuit that it was installed on the lead.

The concentrates discharge from the filters by gravity into railroad cars. In August 1941 the lead concentrate averaged 5.5 per cent moisture and the zinc 5.3 per cent.

The life of the lead filter canvas is about six weeks. It becomes red in color, hard and impervious. Before lime was used in lead flotation the canvas life was only from 6 to 10 days. Lime apparently removes from the lead concentrate some salts that cause the hardening and stopping of the canvas. The zinc filter canvases have a life of 6 to 8 months, but must be washed with dilute muriatic acid about every two weeks, to remove the lime.

Tailing Disposal

The tailings from zinc flotation are pumped to the tailing storage dam by two 3-in. Wilfley pumps in series. One pump formerly handled the tonnage but as mill capacity was increased it was necessary to install another pump about halfway to the dam. The discharge of the first pump is direct-connected to the suction of the second.

Tailings are stored in the conventional manner, by discharging around the outside of the storage pile. The water and the fines run toward a pond in the middle. The excess water, which is clear, is drawn off through a downspout of 4-in. pipe.

TABLE 7.—*Analyses of Flotation Products and Mill Recoveries*

Product	Short Tons	Assays				Recovery or Distribution, Per Cent			
		Ounces per Ton		Per Cent					
		Gold	Silver	Lead	Zinc	Gold	Silver	Lead	Zinc
Heads	95,119	0.013	7.15	7.56	17.30	100.0	100.0	100.0	100.0
Lead concentrate	10,064	0.065	61.32	69.49	5.33	52.5	90.7	97.3	3.3
Zinc concentrate	25,124	0.004	0.90	0.45	60.90	8.8	3.4	1.6	93.0
Tailing	59,931	0.008	0.67	0.14	1.03	38.7	5.9	1.1	3.7

Sampling

Each lot of ore received at the mill is sampled automatically after being crushed. These ore samples are the basis for metallurgical calculations of mill results. A mill-head sample is cut by hand each hour by shutting down the feed conveyor to the rod mill and removing a 6-in. section of ore from the belt. The lead concentrate, zinc concentrate, and two final tailings samples are taken by standard automatic samplers at 15-min. intervals. Samples representing each 8-hr. shift are assayed daily.

Water Supply

Fresh water is piped from the sulphide mines and pumped from a special shaft put

down in a water-bearing area. The alkalinity varies from pH 7.4 to 8.2, being lower in the rainy season.

About one fourth of the mill water is taken from the condenser-water cooling ponds to aid in water treatment at the power plant. Here the alkalinity varies from pH 7.2 to 7.4. The water has been treated with sulphuric acid.

No return water is used in the mill. Water consumption is between 600 and 650 gal. per ton of ore.

Power

A steam turbine power plant is close to the mill. A substation at the mill transforms the power to 440 and 110 volts for mill motors and lighting. Power interruptions are rare.

Power distribution for the month of August 1941 is given in Table 8. The power consumed per ton of ore greatly decreased as the capacity of the mill was increased.

Table 8.—*Power Distribution, August,* 1941

Operation	Kilowatt-hours per Ton of Ore	Percentage of Total Power
Crushing	0.93	5.2
Grinding and classifying	7.12	39.6
Concentration	6.68	37.1
Concentrate disposal	1.19	6.6
Tailing disposal	1.08	6.0
Sampling and assaying	0.44	2.4
Water supply	0.27	1.5
Miscellaneous	0.29	1.6
Total	18.00	100.0

Acknowledgment

This paper is submitted with the approval of the management of Compañia Minera de Peñoles, S. A., to whom the author expresses his thanks. The liberal policy of the management in allowing and encouraging many experiments has brought the mill to its present efficiency. The author is especially grateful for the cooperation received from the local force, both native and foreign, without which present results would not have been possible.

Metallurgy and Milling Practice at Getchell Mine

By Fred Wise,* Member A.I.M.E. and C. W. Wark†

(New York Meeting, February 1940)

Abstract‡

The Getchell mine, a comparatively recent gold discovery, is in the old Potosi mining district, Humboldt County, Nevada. All ore is mined from open pits using Diesel shovels and gasoline trucks. Two types of ore are mined, oxide and sulphide. Oxide ores are siliceous or shaly; sulphide ores either gougy, siliceous or shaly. In mining, the sulphide ore is segregated into hard and gougy ore, because the gouge must be dried before it can be crushed. The oxide ore is not segregated.

General Metallurgy

The oxide ore is the leached outcrop that overlies the sulphide ore body. All of the iron and arsenic sulphides found in the sulphide ores have been thoroughly oxidized. The sulphur and arsenic have been leached out while the iron is present as hematite and limonite. The oxide ore is amenable to direct cyanidation, and although no visible free gold is present it is released at a comparatively coarse grind.

The sulphide ore is more refractory, since the gold is all of micron size and some of it is locked in fine sulphides. It has been established microscopically that the free gold is coated with an impervious shell, which is probably some form of iron compound.

The sulphide ores contain from 1.5 to 2 per cent of arsenic, which is present as orpiment and realgar with minor amounts of arsenopyrite. The orpiment and realgar are of a later mineralization than the gold and are barren.

Roasting and washing of the sulphide ore prior to cyanidation is necessary to eliminate the arsenic, to alter the gold coating from an insoluble form to a soluble form, and to release the gold associated with the fine sulphides. Washing the ground calcine in weak sulphuric acid, trona, or hot water are beneficial in the order named, but to date the water is the best from an economic standpoint.

Metallurgical tests as well as plant practice indicate that a certain amount of gold cannot be recovered even by going beyond economic practices. On this basis metallurgical results are viewed in the light of an optimum tailing rather than a percentage of extraction.

Milling Practice

Crushing (Fig. 1).—There are three primary ore bins, one each for oxide ore, gougy sulphide ore, and hard sulphide ore. In front of each bin a substantial yard is maintained, which makes it possible to stockpile 2500 tons of oxidized ore, 1000 tons of hard sulphide ore, and 1000 tons of gougy sulphide ore. The bins are fed by a 60-hp. caterpillar bulldozer from the stockpiles. Fig. 1 shows the method and

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‡ The entire paper, giving further details and analyses, has been submitted to the American Documentation Institute, 2101 Constitution Avenue, Washington, D. C. It may be obtained from that Institute by ordering Document No. 1444 and remitting 58¢ for copy in microfilm (read enlarged to full size on reading machines now widely available) or $4.00 for copy in paper photoprints legible without mechanical aid.

equipment used in the crushing operation. Oxide ore is crushed to minus ¾ in., and sulphide ore to minus ½ in. The clayey tion to reduce the ore from 12 per cent moisture to 4 per cent is about 1.5 gal. per ton dried.

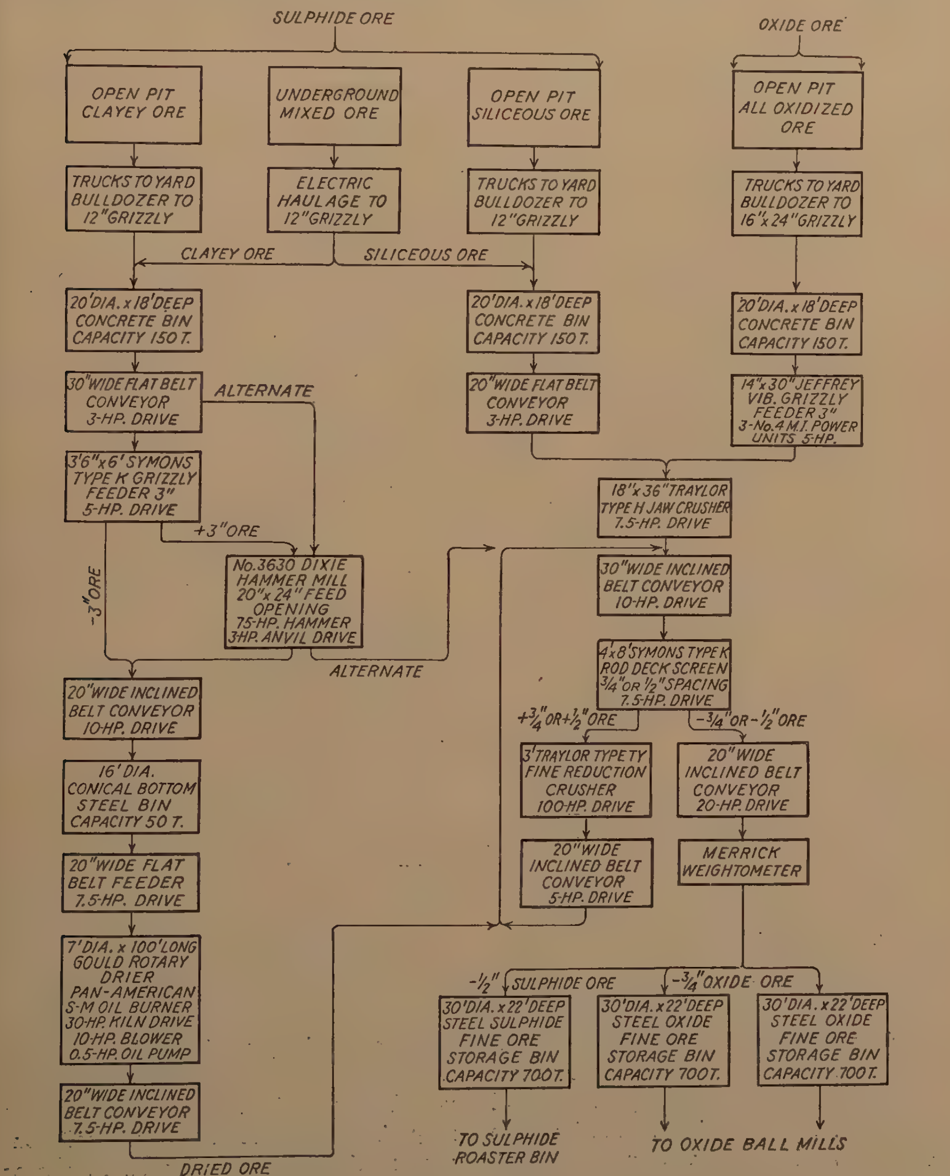

FIG. 1.—FLOWSHEET CRUSHING SECTION, 1000 TONS IN 8 HOURS.

sulphide ore is dried to minus 4 per cent moisture before it can be handled in the convential crushing equipment. Consump-

Oxide Grinding and Classification (Fig. 2).—In the oxide grinding section an average of 600 tons of ore a day is ground.

The unit consists of two 7-ft. by 48-in. Hardinge mills in closed circuit with Dorr classifiers. The rate of feed is controlled by Hardinge Electric Ears and constant-weight feeders, but the dilutions are controlled manually by the operators. Overflow from the mill classifiers goes to a bowl classifier for a sand-slime separation. The ore is ground to minus 20 mesh, and the sand-slime separation is made at 325 mesh (Table 1).

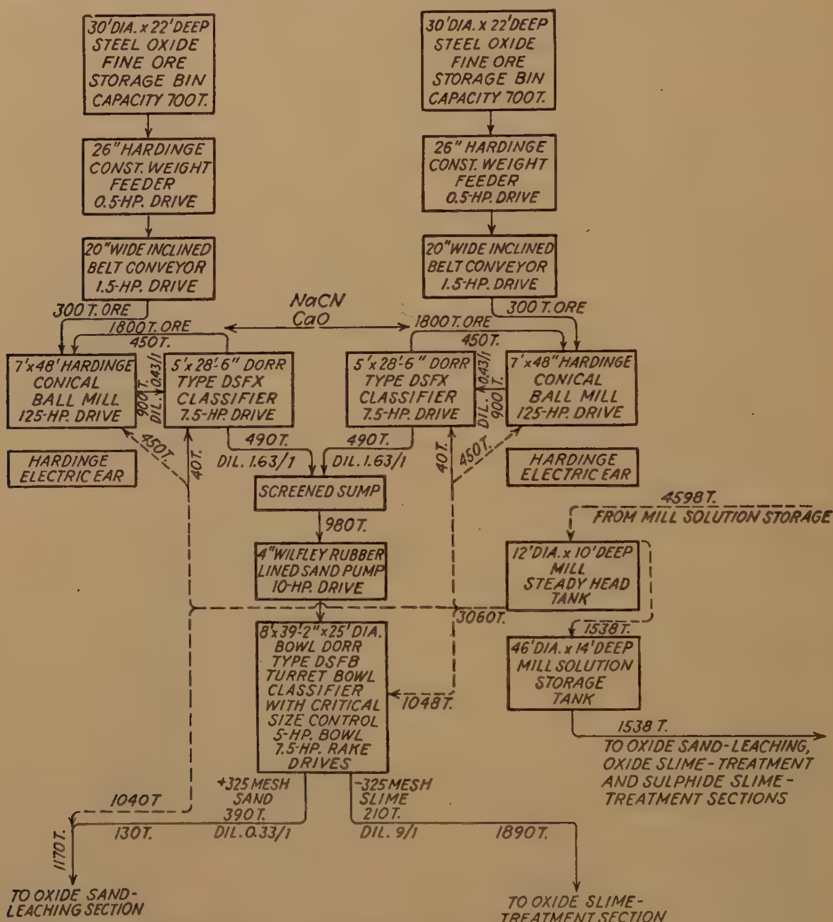

FIG. 2.—OXIDE GRINDING AND CLASSIFICATION SECTION, 600 TONS PER DAY.

From this separation 65 per cent of the ore is taken as a sand product and 35 per cent as a slime product. The ball-mill classifier overflows at 38 per cent solids and the bowl at 10 per cent solids. The mills are operated at 70 per cent solids. Grinding is done in cyanide solution.

Oxide Leaching Section (Fig. 3).—The leaching plant consists of four 46 by 16-ft. vats, each of which holds 1000 tons of dry sand. The vats are filled with solution, charged through a rotating distributor or launder, leached, and discharged through an 8-in. Oliver valve in the bottom to a 5-in. pipe line. A 246-hr. leaching cycle is used, which employs a succession of mill solution, barren solution, and water washes with

drain periods between changes of solution for aeration. A leaching rate of 4 in. per hour is maintained by the use of a V-notch wei metering box. During the first 72 hr. storage. In the final water wash, water is added until the titration of the effluent shows only traces of cyanide.

Oxide Slime-treatment Section (Fig. 4).—

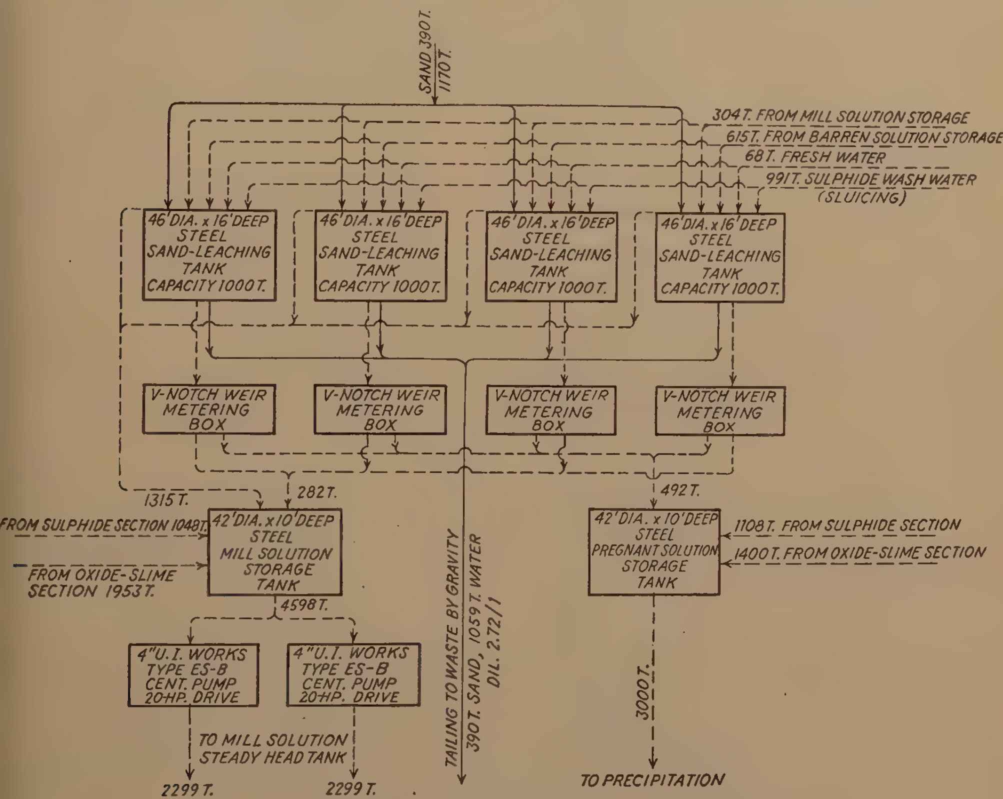

FIG. 3.—OXIDE SAND-LEACHING SECTION, 390 TONS PER DAY.

effluent is sent to precipitation; during the remainder of the time, to mill-solution

TABLE 1.—*Oxide Grinding Screen Analysis*

Ball-mill Feed		Ball-mill Classifier Overflow		Sands, Per Cent Weight	Slimes, Per Cent Weight
Mesh	Per Cent Weight	Mesh	Per Cent Weight		
+3/4	2	+28	6.3	9.7	
3/4–1/2	33	28–35	9.5	14.6	
1/2–3	20	35–48	10.0	15.4	
3–4	15	48–65	10.1	15.6	
4–6	10	65–100	8.3	12.7	
−6	20	100–150	7.5		1.1
		150–200	7.4	20.9	2.6
		200–325	6.8	6.0	8.3
		−325	34.1	5.1	88.0

The oxide slimes are thickened, agitated, washed in a secondary thickener, and then filtered. The material is difficult to handle because the pulp gelatinizes during compression unless it is stirred. The setting of the pulp makes a high underflow density impossible without the formation of islands when a conventional type of thickener mechanism is used. A modified mechanism has partly overcome this difficulty Solution strengths are maintained at 0.5 lb. of cyanide per ton of solution and 0.4 lb. of lime per ton of solution.

Sulphide Roasting and Pretreatment Section (Fig. 5).—Four hundred tons of ore a

day is treated in this section. After crushing to minus ½ in. the ore is roasted in a 260 by 7½-ft. rotary kiln, ground hot in water to 65 mesh, washed, and then cyanided.

The kiln is lined with 6 in. of fire brick backed up by 3 in. of asbestos-block insulation. Dust-collection apparatus consists of a multiclone and a baffled concrete dusthouse. Dust from the multiclone is pulped with water and joined with the ball-mill classifier overflow.

The kiln is fired with fuel oil. About 6½

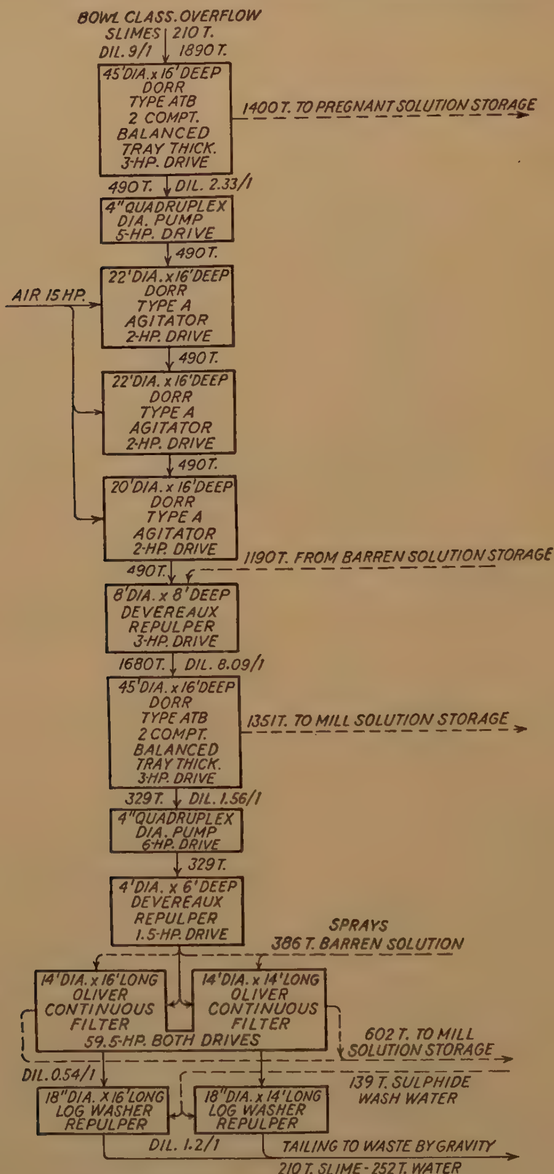

FIG. 4.—OXIDE SLIME-TREATMENT SECTION, 210 TONS PER DAY.

gal. of oil per ton of ore is consumed. A gas velocity of 6 ft. per sec. is maintained

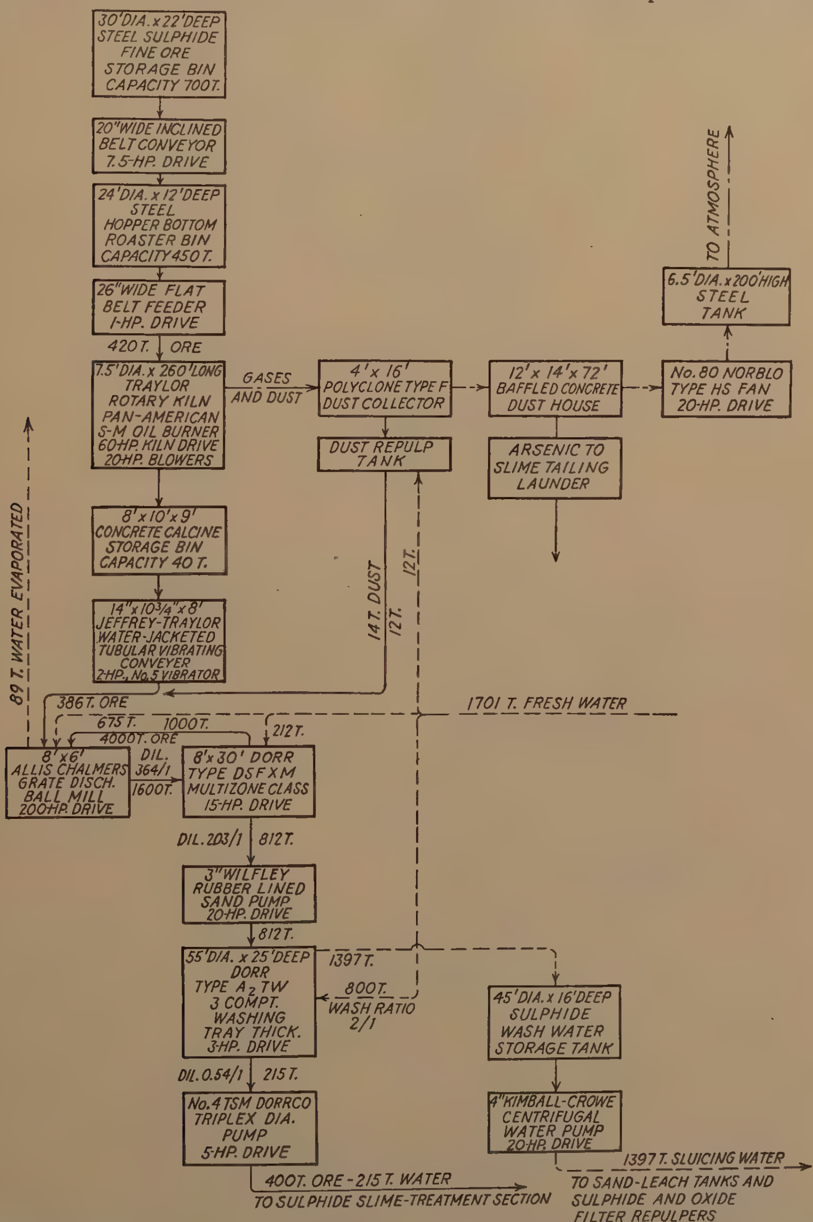

FIG. 5.—SULPHIDE ROASTING, GRINDING AND PRETREATMENT SECTION, 400 TONS PER DAY.

through the kiln. At a kiln speed of 1.25 revolutions per minute it takes the ore 4 hr. to pass through the kiln.

The calcine is stored in a small surge bin, then fed to the scoop box of the ball mill without cooling. No difficulty has been encountered in feeding the hot material although the rock temperature is 1300°F.

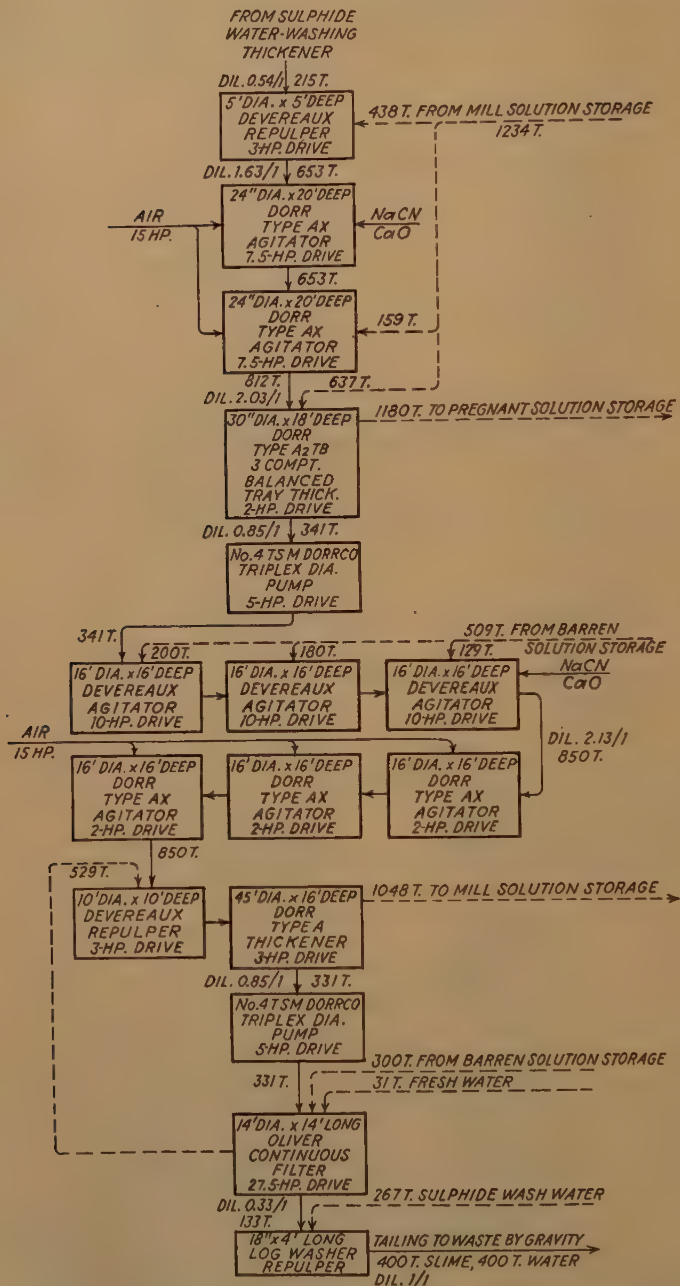

FIG. 6.—SULPHIDE SLIME-TREATMENT SECTION, 400 TONS PER DAY.

The scoop box is vented with a 24-in. stack to allow the steam to escape and the ball-mill bearings are water-jacketed; otherwise the equipment is standard.

Gold extraction is determined by the excellence of oxidation of the sulphide minerals during roasting.

Sulphide Slime Treatment (Fig. 6).—

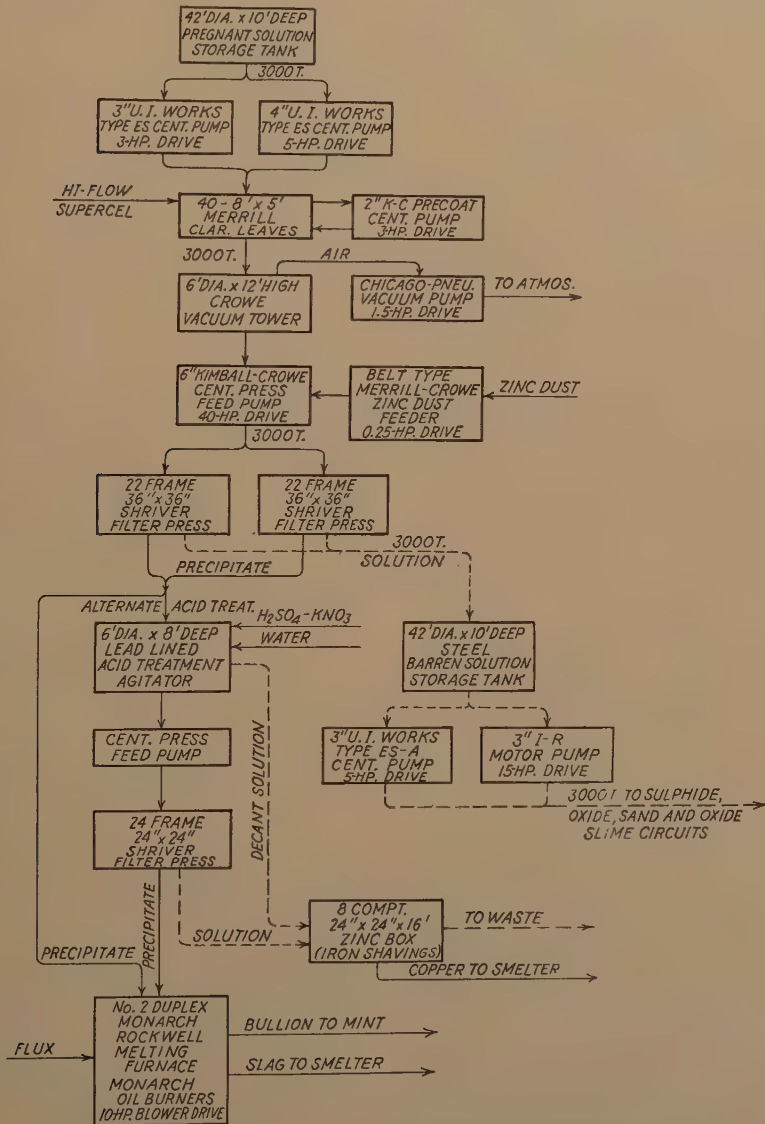

FIG. 7.—PRECIPITATION AND REFINING SECTION, 3000 TONS SOLUTION PER DAY.

The roasting and washing eliminate 90 per cent of the arsenic and 70 per cent of the sulphur. Table 2 shows the operating conditions.

Roasted, ground, and washed sulphide ore is cyanided, using two-stage agitation with a thickening step between stages. After the second agitation the pulp is washed,

thickened and filtered. About 8 hr. of contact is provided in the first agitation circuit and 12 hr. in the second circuit. Solution strengths are held at 0.7 lb. of cyanide per ton of solution and 0.5 lb. of lime per ton of solution. The pulps in this circuit give high underflow densities and only occasional island formation is encountered.

TABLE 2.—*Operating Conditions, Sulphide Roasting and Pretreatment*

Temperatures

Location	Degrees Fahrenheit
Calcine, kiln discharge	1300
Kiln gases, feed end	300
Pulp: ball-mill discharge	180
Classifier overflow	140

Screen Analysis Classifier Overflow

Mesh	Per Cent Weight
+65	1.4
65–100	7.6
100–150	15.2
150–200	12.1
200–325	11.3
−325	52.4

Precipitation and Refining (Fig. 7).—Three thousand tons of solution a day is precipitated, which is a ratio of three parts of so ution to one of ore. A standard Merrill-Crowe simultaneous clarification and precipitation is used. The clarifier leaves are precoated with raw diatomaceous earth, which improves the clarification and thus aids in the general operation of the unit.

In cleaning up, the precipitate presses are blown with compressed air to about 30 per cent moisture, cleaned into trays, and then transferred to the acid treatment tank where 1¼ lb. sulphuric acid per pound of raw dry precipitate is added to the charge, with enough sodium nitrate to convert all of the acid to nitric acid. As soon as the violent action ceases, the charge is diluted with hot water and allowed to agitate for 3 hr. At the end of the agitation period salt is added to precipitate the dissolved silver, and the charge is then filtered and thoroughly washed in a Shriver press.

The acid-treated precipitate is fluxed and melted in two Monarch-Rockwell furnaces in the conventional manner. Most of the copper and zinc is removed in the acid treatment, thus a bullion from 900 to 950 fine in gold is obtained with a very low flux ratio. The furnaces are lined with a sillimanite ramming mix, which also aids in the production of a matte-free high-grade bullion.

The following flux formula is used: raw precipitate 100 per cent; silica, 5; borax, 10; soda ash, 15; niter, 5.

Table 3 shows general operating data.

TABLE 3.—*General Operating Data*

WATER BALANCE

Water In	Tons per Day	Water Out	Tons per Day
Sulphide pretreatment	1,701	Sulphide tailing	133
Sulph de filter	31	Sulphide sluicing	267
Sand leaching	34	Oxide sand tailing	34
Sand sluicing	34	Oxide sand sluicing	1,025
		Oxide slime tailing	113
		Oxide slime sluicing	139
		Evaporation sulphide	89
Total	1,800		1,800

CHEMICAL CONSUMPTION, LB. PER TON

Ore	Sodium Cyanide	Lime (CaO)	Zinc	Lead Nitrate
Oxide ore	0.50	8.60	0.09	0.11
Sulphide ore	1.10			
Solution precipitated			0.03	0.003

ELECTRIC POWER DISTRIBUTION

Operation	Kw-hr. per Ton Milled	Percentage of Total
Crushing	2.32	10.52
Roasting	1.88	8.56
Oxide grinding	5.07	23.02
Sulphide grinding	2.61	11.86
Sulphide pretreatment	3.55	16.11
Slime treatment	2.84	12.91
Sand leaching	0.52	02.35
Precipitation and refining	0.99	04.48
Freh water	2.24	10.19
Total	22.02	100.00

LABOR DISTRIBUTION

Item	Tons Ore per Man-shift	Man-hours per Ton Ore
Mining	154.0	0.052
Milling	33.3	0.24

Developments in the Concentrating of Minnesota Iron Ores

By T. B. Counselman,* Member A.I.M.E.

(Duluth Meeting, August 1941)

The importance of concentration of iron ores too low in grade to be smelted direct is shown by Table 1, showing 1940 shipments from the Lake Superior district. Canadian ores are omitted.

Table 1.—*Shipments from Lake Superior District in 1940*[a]

Shipments	Gross Tons	Percentage of Total Shipments	Percentage of Minnesota Shipments
Total shipments	63,948,846	100.0	
Total beneficiated[b]	29,813,990	46.7	
Total shipments from Minnesota	48,949,322	76.5	100.0
Total from Minnesota beneficiated[b]	25,551,722		52.2
Total from Minnesota concentrated[c]	9,213,380	14.4	18.8

[a] Data from Lake Superior Iron Ore Association.
[b] Includes crushing and screening.
[c] Includes sintering and drying.

Table 2.—*Tonnage Produced in Minnesota*

Product	Gross Tons	Per Cent
Washed	7,233,651	78.6
Jigged	1,114,904	12.1
High density	380,872	4.1
Total concentrates by wet methods	8,729,427	94.8
Sintered	222,710	2.4
Dried	261,243	2.8
Total concentrates	9,213,380	100.0

Only in Minnesota is concentrating, other than crushing and screening, practiced at present. Tonnage produced by various methods of concentration is shown in Table 2.

The curves in Fig. 1 show the gradual increase in tonnage of concentrated ores shipped, compared with total shipments from Minnesota.

Most of the concentration work is performed on the Mesabi Range. This range is about 100 miles long, and not over 3 miles wide from the line at which the formation tapers out to a feather edge on the north to the place on the south at which, some 600 ft. thick, it is overlain with slates.

Classes of Ore

The Mesabi Range contains three classes of iron ore. First is the unaltered taconite, or banded iron-bearing chert. It is estimated that there are some 60 billion tons of this material, which contains, roughly, 25 to 30 per cent iron. This material is virtually untouched, since no commercially successful method has yet been found for concentrating it to a grade suitable for the blast furnace. Intensive experimental work is in progress, and the problem is gradually being solved. In 1922 the Mesabi Iron Co. spent several million dollars erecting a large-scale pilot plant for concentrating and sintering the magnetic taconites on the east end of the range. An important tonnage of high-grade sinter was shipped to lower lake furnaces, proving that the taconite can be concentrated to a satisfactory grade. The costs, with equipment and methods then in use, were too high for

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* Manager, Industrial Division, The Dorr Company, Inc., Chicago, Ill.

the product to be competitive with other iron ores.

The second class consists of the merchantable or direct shipping ores. These were formed by cracking and folding of the taconite, with subsequent leaching out of the silica, leaving iron oxide behind. All of the big open pits and underground mines are operating on such concentrations.

The third class of ore lies between the first two. The silica has been loosened from the iron in varying degree, but has been only partly removed by leaching. These are the so-called sandy or wash ores of the western Mesabi, and it is on these ores that concentration is practiced.

This third class of ore is subdivided into several different classes. There is first the easily washed ore, which consists of coarse and fine lumps of high-grade iron oxide mixed with fine free silica, which can be removed by rinsing the ore in a stream of water or sizing it at 60 or 80 mesh. In other classes of this wash ore, not only do the silica particles become coarser, but the lumps of iron oxide become more siliceous and approach closer and closer to unaltered taconite. Thus on the poorer-grade ores, jigging or some other form of treatment becomes necessary.

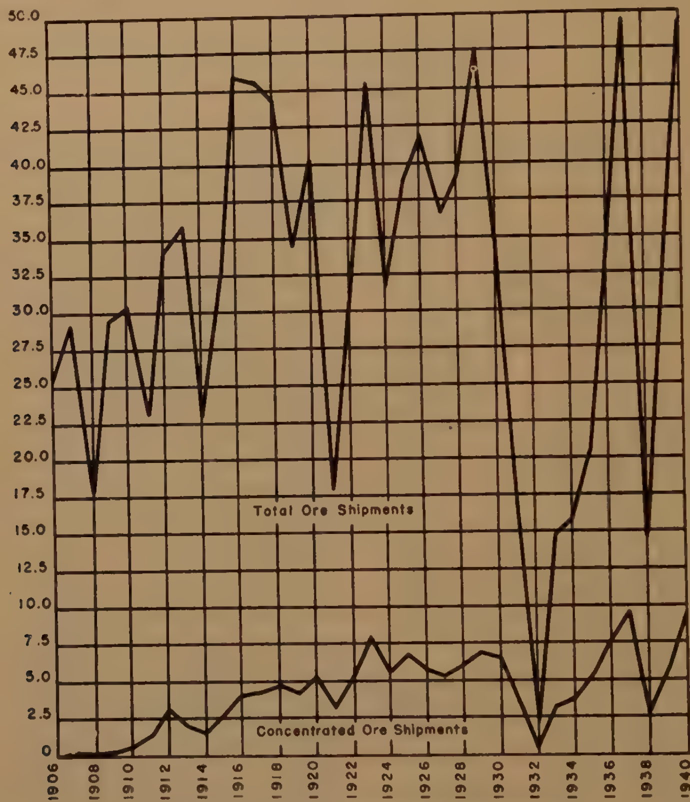

FIG. 1.—IRON-ORE SHIPMENTS, MINNESOTA.
From Mining Directory of Minnesota Mines Experiment Station, University of Minnesota.

Concentrating Wash Ore

The first attempts to concentrate the wash ore were made as early as 1901, and the first commercial plant operated during the 1910 season. The flowsheet of the

plant is shown in Fig. 2 and the results of the first season's operation in Table 3.*

TABLE 3.—*Results of Operation, One Plant*

	First Season	1940 Season
Tons crude ore treated	950,000	2,600,000
Tons crude ore per hour per unit	75	660
Tons concentrate produced	610,000	1,580,000
Tons concentrate per hour per unit		400
Weight recovery, approx., per cent	65	61
Crude ore: iron, per cent	45.5	44.00
Phosphorus, per cent	0.053	0.036
Concentrate: iron, per cent	56.60	56.95
Phosphorus, per cent	0.058	0.047
Iron unit recovery, per cent	80.0	79

Through the years the changes that have taken place in this flowsheet have been characteristic of the basic changes in practice of washing straight wash ores. The flowsheet has been simplified, finer crushing has been found necessary, and capacities have been increased, until the present flowsheet is somewhat as shown in Fig. 3. The same general principles apply as in the earlier flowsheet. The results obtained in the same plant during the 1940 operating season are shown in Table 3. The results for the two years are similar but with the more difficult ore now being treated and the much higher tonnage per unit, metallurgical results are very good.

At another plant, with an almost identical flowsheet, which started operations about 1920 with a capacity of about 150 tons per hour per unit, and where secondary crushing and additional classification have brought the capacity up to about 375 tons per hour per unit, the average results for the 1940 season were as shown in Table 4.

TABLE 4.—*Operating Results, Second Plant*

	Per Cent
Crude ore: iron	45.42
Silica	28.61
Concentrate: iron	59.38
Silica	7.18
Phosphorus	0.047
Moisture	7.97
Tailing iron analysis, approx	15.0
Weight recovery	69.38
Iron unit recovery	90.73

EARLY JIG PRACTICE

All of the results given in the preceding paragraphs are for straight wash ore. Secondary crushing was employed, but no jigging or similar concentration. As early as 1924 jigging was practiced, and has been in continuous operation ever since. At many properties the wash ore is of such poor grade that in no other way can a satisfactory shipping product be obtained.

Early jigging was with McLanahan-Stone jigs treating sized feeds down to ¼ in. Of course, only a part of the total tonnage of crude ore was coarser than this size. Typical results of a season's operation of a jigging plant operating under these conditions are given in Table 5.

TABLE 5.—*Typical Results of Jigging*

	Per Cent
Jig feed: iron	53.18
Silica	16.56
Concentrate: iron	57.94
Silica	10.24
Tailing: iron	42.31
Silica	31.01
Weight recovery, jigs only	69.6
Iron unit recovery, jigs only	75.7
Percentage total concentrate produced by jigs	49.3
Over-all weight recovery from crude ore	40.3

At first glance, the tailing looks altogether too high in iron content, but the fact remains that the jigs took as feed an unsalable ore and recovered nearly 70 per cent of it as a marketable product. The jig tailing was stockpiled and has now been all reworked, as will be discussed later.

RHEOLAVEUR

The material finer than ¼ in. on these particular ores could not be brought up to

* It should be noted throughout this paper that the figures given in one table cannot be compared with those given in another, because of the wide variation in the characteristics of the ores being treated, not only from different open pits but from the same pit over a period of years. The pits usually have some direct shipping ore, surrounded by an increasingly poorer grade of wash ore, finally approaching unaltered taconite.

a satisfactory grade by classification, as practiced on straight wash ores. Rheolaveur plants, following coal-preparation practice, were installed at two washing plants. This

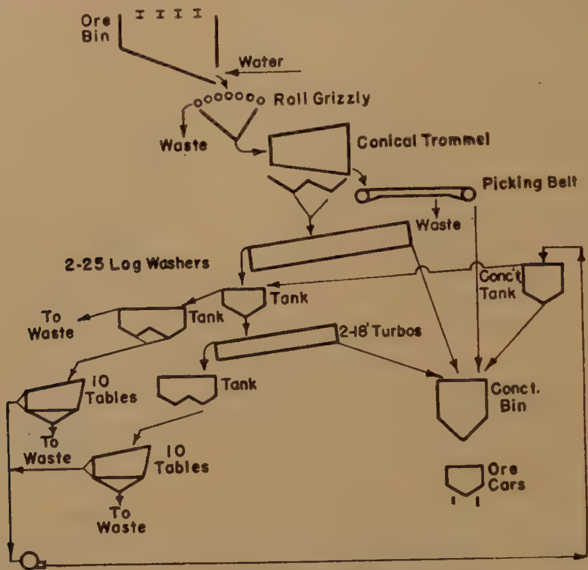

FIG. 2.—ORIGINAL MESABI FLOWSHEET FOR WASHING IRON ORES.

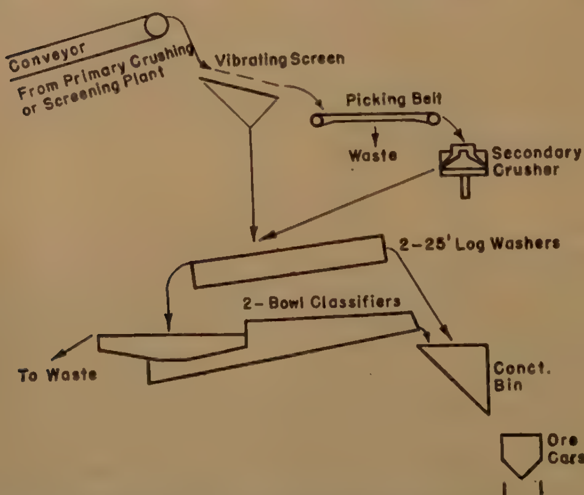

FIG. 3.—GENERALIZED FLOWSHEET FOR WASHING MESABI ORES, WITH BOWL CLASSIFIERS AND SECONDARY CRUSHING.

method of treatment involves repeated launder classification. Typical work for a season is shown in Table 6.

Note that the concentrate has not been brought down to the desired figure of 12 per cent silica. This, together with the small capacity and the large amount of labor and water required, has led to the gradual replacement of these units by jigs capable of treating material of this size.

TABLE 6.—*Typical Rheolaveur Work*

	PER CENT
Feed to Rheolaveur plant: iron	51.83
Silica	19.45
Concentrate: iron	55.20
Silica	14.72
Tailing: iron	49.54
Silica	24.64
Weight recovery (Rheolaveur only)	40.5
Iron unit recovery (Rheolaveur only)	43.1

MAGNETIC ROASTING AND CONCENTRATION

One of the proposed methods of treating unaltered taconite is to give it a magnetizing roast, then grind and concentrate magnetically. With this objective in view, a shaft type of roasting furnace was designed at the Mines Experiment Station at the University of Minnesota, and experimental tests were made on samples of taconite from many localities.

The tailing from the McLanahan-Stone jigs, shown in Table 5, seemed to be ideal raw material for such treatment. It was mined, crushed to just about proper size, the fines had been screened out, and small-scale tests revealed that it should respond well to the treatment.

Therefore, in 1934, under a joint project between the Mines Experiment Station and one of the operating companies, a 10-ton-per-hour pilot plant was built on the range and operated for five seasons, long enough to give all the necessary factors for plant design and cost estimates. The results of this work have all been published.[1] The flowsheet of this pilot plant is shown in Fig. 4 and the results for the 1937 season are given in Table 7.

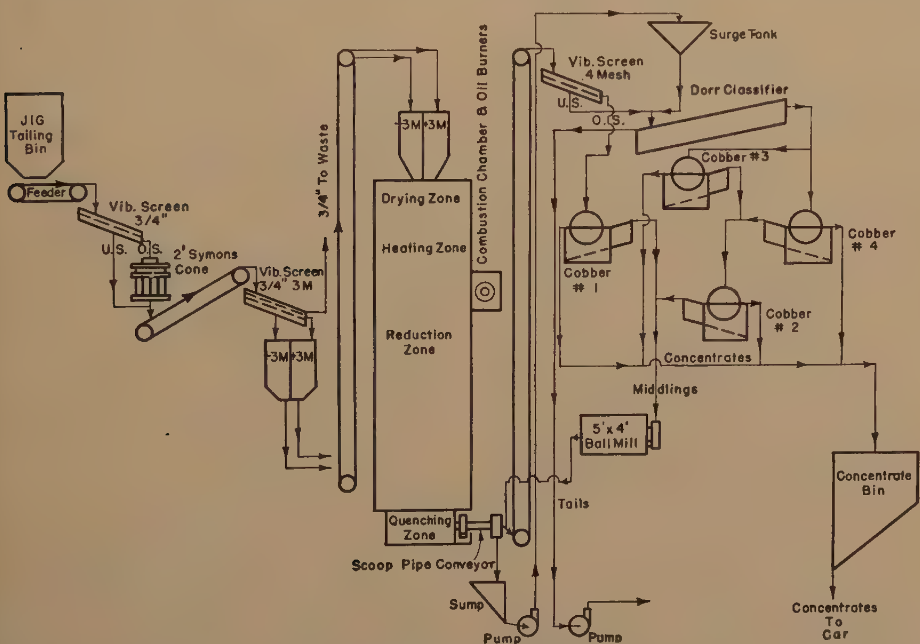

FIG. 4.—FLOWSHEET OF MAGNETIC ROASTING AND CONCENTRATING PLANT OPERATING ON JIG TAILINGS, MESABI RANGE.

TABLE 7.—*Pilot-plant Results*

	PER CENT
Jig tailing fed to magnetic plant: iron	46.07
Silica	28.02
Product of reducing furnace: iron	51.54
Silica	25.54
Concentrate: iron	61.79
Silica	12.02
Tailing: iron	21.73
Weight recovery	63.75
Iron unit recovery	89.3

[1] References are at the end of the paper.

The furnace, of course, drives off any water of crystallization from limonite, and takes away one atom of oxygen from hematite, leaving magnetite.

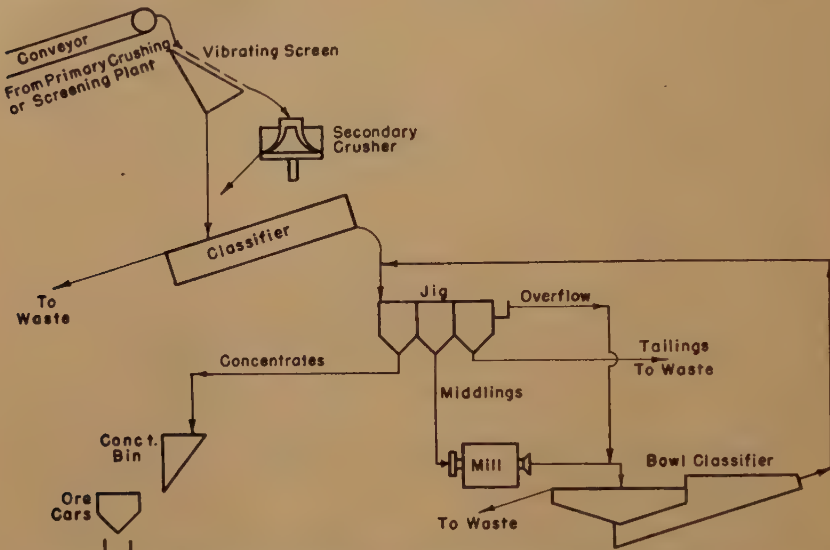

FIG. 5.—TYPICAL SIMPLIFIED FLOWSHEET OF CURRENT PRACTICE OF JIGGING MESABI IRON ORES

The iron lost in the tailing (Table 7) was virtually all nonmagnetic, and was almost all in the silicate form, therefore not recoverable.

MODERN JIG PRACTICE

Intensive development of jigging practice has been in progress for many years.

TABLE 8.—*Typical Jigging Results*

	PER CENT
Crude ore: iron	40.00
Silica[a]	37.50 to 39.50
Phosphorus	0.027
Moisture	7.30
Concentrate: iron	57.50
Silica[a]	12.00 to 14.00
Phosphorus	0.34
Moisture	5.00
Tailing: iron	24.00
Silica[a]	61.00 to 63.00
Weight recovery	48.00
Iron unit recovery	69.00

[a] The variations in silica analyses are caused by different hematite-limonite ratios in the ore.

Standard jigs are intended for a relatively small quantity of heavy product, concentrate in the case of base-metal ores, or refuse in the case of coal. With iron ore, however, the heavy product is the largest part of the feed, 40 to 65 per cent or more. Consequently, jigs had to be modified and new jigs developed to meet these conditions. Also, it was necessary to develop the technique of jigging so as to be able to handle an unsized feed.

This has been accomplished, and there are now several different jigs in operation that meet the required conditions. A typical flowsheet is shown in Fig. 5, and operating results in Table 8.

JIGGING, VERMILION RANGE

The Vermilion Range, with the Soudan mine near Tower and four mines at Ely, has always been thought of as a producer of high-grade hard ores, no concentration being required. A few years ago the Chandler mine was exhausted as far as direct shipping ore was concerned. There was still some ore in the pillars, which would be contaminated by capping if these were caved, and there were also some lean-ore stockpiles in the old pit.

Before the 1940 season, a jig plant was built to treat these lean-ore stockpiles and

underground ore from the caved pillars. During the 1940 season the results shown in Table 9 were obtained.

TABLE 9.—*Jigging Results*, 1940

Mixture of underground and stockpile ore, tons	189,644
Iron analysis, per cent	38.96
Silica analysis, per cent	24.19
Phosphorus analysis, per cent	0.046
Concentrate, tons	74,426
Iron analysis, per cent	53.21
Silica analysis, per cent	12.66
Phosphorus analysis, per cent	0.040
Moisture, per cent	4.05
Apparent weight recovery, per cent	39.25
From underground ore: weight recovery, per cent	50
Silica in concentrate, per cent	10
From stockpile ore: weight recovery, per cent	30
Silica in concentrate, per cent	14

This is certainly an interesting operation, indicating the possibility of making an additional recovery, in hard-ore mines, by jigging the caved pillars when the mine is worked out, and also by jigging lean ore removed during mining operation. Jigs were used many years ago on some of the Michigan ores, so that the Chandler operation cannot be called the first application of jigs to hard ores, but certainly it is the most recent and the only such application in operation at the present time.

Differential Density

Since 1937 work has been under way on sink-and-float separations, using heavy fluids. Since it was necessary for the fluid to have a gravity of approximately 3.20, no true solution of commercial practicability was available, and such fluids have been obtained by using thick concentrations of fine solids suspended in water. In principle, this is analogous to the Chance process of coal cleaning, where sand is used as a medium.

In order to obtain the desired gravity, galena flotation concentrate was first used as a medium, this being in use in two commercial plants.*

* American Zinc Co., Mascot, Tenn.; Eagle Picher Mining and Smelting Co., Picher, Oklahoma.

Galena medium gave attractive results, so far as the actual separation of iron ore from coarse gangue was concerned, but the cleaning of the galena medium proved altogether too complicated under the conditions involved. Therefore, after considerable intensive experimental work, ferrosilicon crushed to about 65 mesh was adopted as a medium, and was found to be entirely satisfactory. Ferrosilicon can be cleaned with magnetic separators without great difficulty.

This process has been written up in two excellent papers.[2,3] To date, it has been found preferable to treat plus ¼-in. material by this method, the undersize of the ¼-in. screens being treated on suitable jigs. Efforts are being made to handle finer material and undoubtedly will be successful down to some limiting size below which the viscosity of the fluid will not permit a satisfactory separation.

A considerable tonnage of the McLanahan-Stone jig tailing, shown in Table 5, was retreated by this differential density process and the results (Table 10) may be compared with the results obtained by magnetic roasting and concentration (Table 7).

TABLE 10.—*Results of Differential Density Method*

	Per Cent
Jig tailing, fed to DD plant: iron	46.28
Silica	27.62
Concentrate: iron	57.08
Silica	12.14
Tailing: iron	33.69
Silica	45.67
Weight recovery	53.8
Iron unit recovery	66.5

The differential density process is also in operation on Mesabi so-called jig ores. A differential density flowsheet is shown in Fig. 6, and results for a season's operation are given in Table 11. The slimes and fine sandy tailing were eliminated from the crude ore before the partly concentrated material was divided between the differential density and jig plants (by ¼-in. screening).

Flotation

Flotation tests were made as early as 1916, in a desultory way, on Minnesota iron ores. Tests have been conducted from time to time in the ore-dressing laboratory of the University of Minnesota on flotation of fine iron from washing-plant tailing, and these results have been published.[4,5]

There seem to be some advantages in treating current washing-plant tailings by flotation. It is hardly likely that as yet it would pay to do much grinding of lean ore and taconite to flotation size, because magnetic concentration, after a reducing roast, can be conducted at a much coarser grind than flotation. But for washing-plant tailing, which requires no grinding and no roasting, flotation has excellent possibilities. The results to date have been summarized in a recent paper.[6]

Considerable work has been done with flotation on the manganiferous ores of the Cuyuna Range also. Following experimental work at the Bureau of Mines at Rolla, Mo., a 10-ton-per-hour pilot plant including tabling and flotation was operated for a season on the Cuyuna Range, in an effort to concentrate the black manganiferous ores to spiegel or even

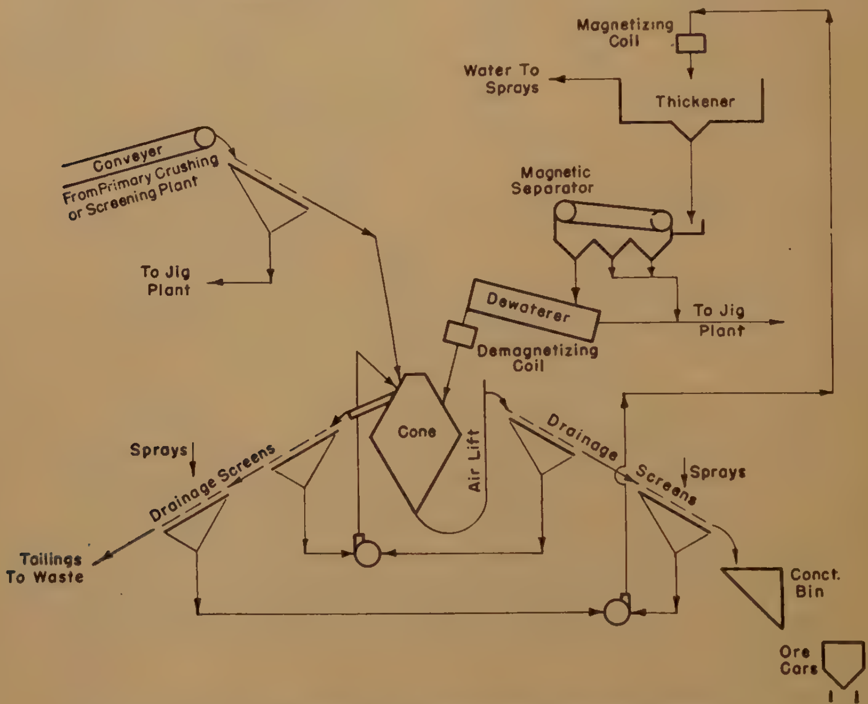

FIG. 6.—FLOWSHEET OF A DIFFERENTIAL DENSITY PLANT.

TABLE 11.—*Season's Operations*

	Differential Density, Minus 1¼ + ¼-in., Per Cent	Jig, Minus ¼-in., Per Cent
Washed feed: iron	53.18	51.49
Silica	18.32	19.72
Concentrate: iron	59.88	56.29
Silica	8.76	13.05
Tailing: iron	33.73	46.63
Silica	46.11	26.55
Weight recovery	74.4	50.5
Iron unit recovery	83.8	55.1
Proportion of total concentrate produced	56.6	43.4
Over-all weight recovery from ore	40.0	

concentrate of ferromanganese grade. Flotation proved unreliable because of the varying amount of carbonate (rhodochrosite) present, which would vary, uncontrollably, the grade of flotation concentrate.

More recently, a very thorough investigation was made by the Bureau, using a flowsheet embodying jigging, fine grinding, and bulk flotation of iron and manganese, magnetic roasting of the flotation concentrate, and magnetic separation to obtain an iron product and a high manganese product.[7]

Simplified Washing Plants

For relatively small tonnages of wash ores, where the customary large and expensive washing plants cannot be justified, a simplified inexpensive washing plant has lately reappeared. These plants consist essentially of a screen and a classifier. Such a plant was used at the Margaret property years ago, and two more such plants have recently been built at the Argonne and Galbraith properties. The latter plant has recently been written up in Skillings' *Mining Review* for June 7, 1941. Everything is kept close to the ground, with extensive use of conveyors for transfer of material between steps of concentration, and the structures are temporary and inexpensive, rather than permanent and costly.

Sintering and Drying

Iron ores analyzing below 50 per cent Fe natural are penalized by a decreased price per unit of iron. Many ores contain enough free moisture to put them in a penalty class. This can be corrected by partial drying and there is a drying plant on the Cuyuna Range. Other ores are limonitic and contain water of crystallization. This can be driven off by sintering, and there is also a sintering plant on the Cuyuna Range. Formerly there was a sintering plant on the Mesabi Range also, at Chisholm, but it has been moved away.

Aside from moisture removal, sintering is the most widely practiced form of agglomeration. Material so fine that it would largely blow out as flue dust if charged into the furnace can be agglomerated by sintering. The sinter is a highly desirable material for the blast furnace and speeds up its operation.

Sintering at the head of the lakes, however, is very costly, because of the expense of carbon for fuel, oil for ignition, and the additional overhead due to five months' enforced idleness while navigation is closed. Harrison[8] states that sintering for agglomeration only costs about $1.20 at the head of the lakes, against about $0.60 at the furnace plant.

On the other hand, with a limonitic ore that would fall in the penalty class, sintering in Minnesota can be a very paying proposition even at a high cost, because of a higher price for the ore and a lower freight charge. Each ore must be figured out on its own merits.

Table 12 gives average results for 1940 on three Cuyuna ores.

Cuyuna Range as Source of Manganese

There is an important tonnage of manganiferous iron ore on the Cuyuna Range. Attempts to concentrate the black ores by mineral-dressing methods have not been successful. Wilson Bradley did a great deal of work some years ago on leaching with ammonium sulphate and consideration of this method has lately been revived. There has also been some talk of SO_2 leaching of this black ore.

The brown ores are now being concentrated by differential density, and it is reported that a concentrate of spiegel grade is being produced.

Further activity in production of manganese from this district may be expected,

if we are to become self-supporting with respect to manganese.

The foregoing has been a rather complete record of the various improvements made in concentration methods since the beginning of these operations. Possibly the more recent of these improvements may be summarized, in chronological order, as follows:

1. Magnetic roasting and concentration.

2. Development of several improved jigs and improved jigging practice.

3. Differential density with ferrosilicon medium.

4. Application of jigging to hard ores.

5. Simplified and less expensive plants, for relatively small tonnages—plants that can be moved when desired.

TABLE 12.—*Average Results on Three Cuyuna Ores*, 1940

PER CENT

	Ore A		Ore B		Ore C	
	Before Sintering	After Sintering	Before Sintering	After Sintering	Before Sintering	After Sintering
Iron analysis	44.27	56.59	41.59	55.62	33.08	45.44
Phosphorus analysis	0.218	0.270	0.221	0.297	0.193	0.301
Manganese analysis	1.42	2.37	1.95	2.92	7.70	10.59
Silica analysis	7.36	10.70	7.67	11.07	10.17	14.44
Loss on ignition	6.70	2.89[a]	9.21	2.56[a]	6.71	2.18[a]
Moisture (free)	16.62	0.81	18.80	1.01	18.60	0.97
Weight recovery		75.2		72.9		70.3

[a] Gain on ignition.

ACKNOWLEDGMENTS

Thanks are due the several operating companies for permission to publish the figures given herein. The data have purposely not been identified as to plant or company, therefore this acknowledgment is also kept in general terms, but the close and friendly cooperation of the principal officials of most of the operating companies is gratefully acknowledged.

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Launders

By Harold A. Linke*

(New York Meeting, February 1940)

The following article presents notes and data compiled and computed by the writer for use in the determination of: size and slope of mill launders, details of junction boxes and downspouts, and distribution of pulp. It also includes pulp formulas, volume of discharge through bushings, paths of discharging streams, banking of turns and curves.

In the design of a concentrating mill, the draftsman usually considers launders to the extent of providing ample "head" for transmission of pulp from one department of the mill to another. The superintendent of construction aims to build his launders wide and deep, and on a slope steep enough to ensure positive flow. So launders are laid generally on too steep a slope, with the result that the repair gang works to maintain launder bottoms and sides against the erosive action of the conveyed pulp. To this end many kinds of liners have been devised: concrete, mastic, white iron, glass, rubber.

True, the reduction of slope to the point where a pulp will flow with minimum wear, will minimize labor and expense; but when the desired slope is not predetermined, the millman hesitates to readjust his launder slopes by cut-and-try methods.

To the end that the desired size and slope may be found readily, the writer has: (1) conducted experiments with pulps of different specific gravity, screen analysis and dilution; (2) gathered a few graphic data from standard reference works (such as Peele's Mining Engineer's Handbook and Taggart's Handbook of Ore Dressing); (3) computed several tables; and compiled results that have proved dependable for Utah Copper ore. It is quite possible that the following tables and graphs may not be applicable to all ores, but it is hoped that they may prove helpful, even in slight degree, in alleviating a portion of the trouble and migraine of some other millman.

Launder Slopes

The main object in the study of launder slopes is to provide grade steep enough to prevent "sanding" by a safe margin, and to avoid

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* Utah Copper Co., Garfield, Utah.

excessive grade, which causes undue wear of launder bottom and sides. Three factors enter into the calculation: (1) size of particles in the pulp; (2) percentage of solids, and (3) specific gravity of the dry ore. It is obvious that less slope is necessary if the ore is finely ground, if the dilution is great, and if the ore is light than if the ore were coarse and heavy and the pulp thick.

A pulp is considered with regard to all of these factors and the launder (always of rectangular cross section) is graded to suit the steepest slope required by any of them.

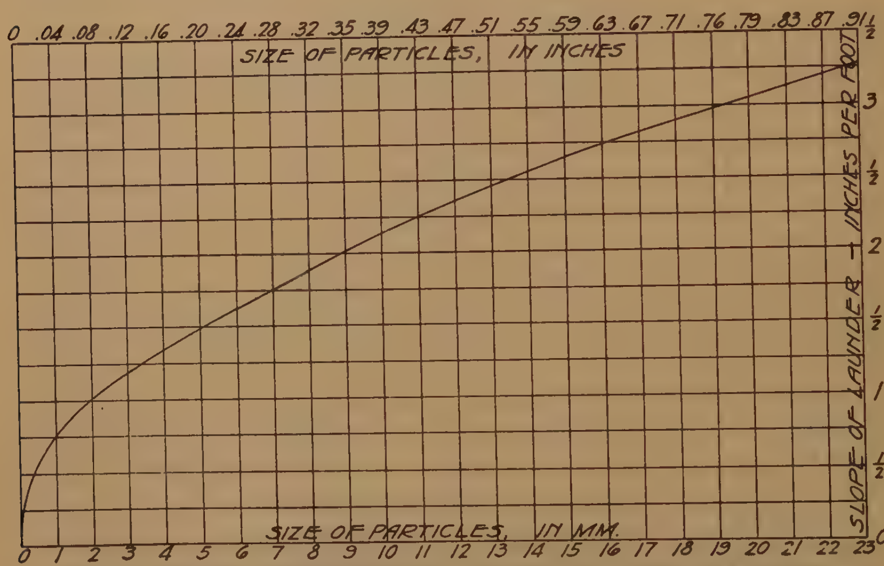

Fig. 1.—Relation between size of particles and slope of launder.
Copy of graph given by Taggart, p. 1091 of Handbook of Ore Dressing, to which the writer has added the curve to the right of 13 mm. The curve has been found, by trial, to be satisfactory for use with Utah Copper Company ore.

Table 1.—*Tyler Standard Screen Sieves*

Mesh	Size of Opening		Mesh	Size of Opening	
	Inches	Mm.		Inches	Mm.
	1.050	26.67	14	0.046	1.168
	0.742	18.85	20	0.0328	0.833
	0.525	13.33	28	0.0232	0.589
	0.371	9.423	35	0.0164	0.417
3	0.263	9.680	48	0.0116	0.295
4	0.185	4.699	65	0.0082	0.208
6	0.131	3.327	100	0.0058	0.147
8	0.093	2.362	150	0.0041	0.104
10	0.065	1.651	200	0.0029	0.074

Slope of Launder as Governed by Size of Grains.—In the determination of the slope on which to construct a launder to carry a given pulp, the first step is to ascertain the slope required by the largest ore particles in that pulp. Table 1 gives the maximum sizes of particles, in inches and millimeters, that will pass through standard screen sieves. For example: An ore ground to pass through a standard 35-mesh screen is found, by reference to Table 1, to be minus 0.0164 in. (or minus 0.417 mm.). Fig. 1 shows that, for particles of this size, a slope of ½ in. per foot is required.

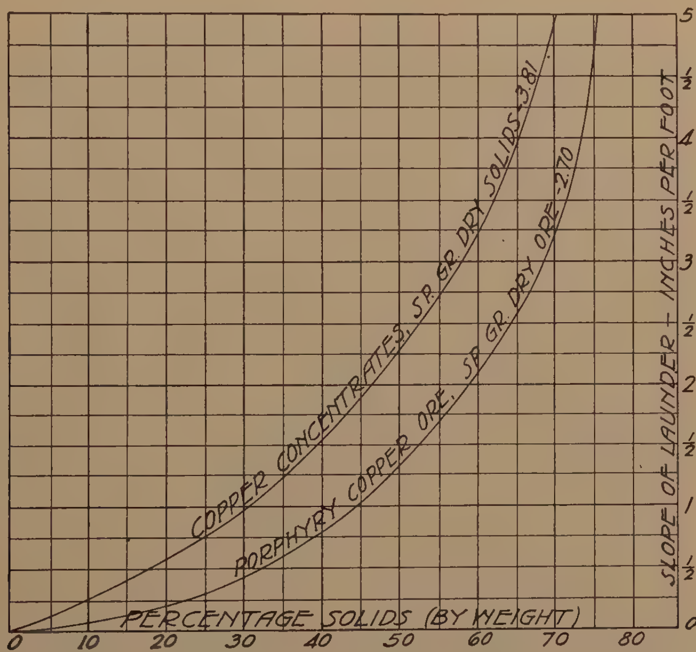

Fig. 2.—Relation between percentage of solids and slope of launder.

Slope of Launder as Governed by Percentage of Solids and Specific Gravity.—The next step is to determine the safe slope required to carry a pulp, based on the percentage of solids of the pulp and the specific gravity of the dry one. Fig. 2 shows the relation between percentage of solids (by weight) and the required launder slope to carry two classes of material whose specific gravities are given. For example: Consider the ore mentioned in the previous example. The specific gravity of the dry ore is 2.7 and the percentage of solids (by weight) is 45. Fig. 2 shows that this pulp requires a launder slope of 1 in. per foot. This slope, being greater than that determined by size of particles, must govern; and it is the slope on which the launder must be laid.

Size of Launders

Shape of Cross Section.—As clearly stated by Peele: "frictional resistance can be reduced in flumes or conduits carrying water by making the channel such that the wetted perimeter is the smallest possible for the volume flowing. This is accomplished by making the bottom semicircular." In transporting pulps, however, where part of the solids is carried by rolling, "a semicircular bottom restricts the area available for free motion of the particles, and a rectangular cross section is better." In launder construction the rectangular cross section only is used.

TABLE 2 — FLOW OF WATER IN RECTANGULAR LAUNDERS — COMPUTED BY H.A.LINKE

SLOPE (INCHES PER FOOT)		1/8"		1/4"		1/2"		1"		2"		3"		4"	
WIDTH OF LAUNDER IN INCHES	DEPTH OF WATER IN INCHES	GALLONS PER MIN	VELOCITY FT/MIN	GALLONS PER MIN	VELOCITY FT/MIN	GALLONS PER MIN	VELOCITY FT/MIN	GALLONS PER MIN	VELOCITY FT/MIN	GALLONS PER MIN	VELOCITY FT/MIN	GALLONS PER MIN	VELOCITY FT/MIN	GALLONS PER MIN	VELOCITY FT/MIN
4"	1/2"	6.0	57.6	8.5	81.5	12.0	115.2	16.9	162.7	23.8	229.0	28.9	278.1	33.0	318
	1"	18.5	88.9	26.1	125.7	36.9	177.7	52.1	251.0	73.4	353.0	89.2	429.0	101.8	490
	2"	51.7	124.5	73.1	176.0	103.4	248.8	146.0	351.4	205.5	494.4	249.6	600.5	284.9	686.
	3"	90.3	144.9	127.7	204.8	180.5	289.6	255.0	409.0	358.7	575.4	435.7	698.9	497.5	798.
	4"	130.8	157.4	185.0	222.5	261.5	314.6	369.4	444.4	519.6	625.1	631.2	759.4	720.8	867
8"	1"	43.1	103.6	61.0	146.7	86.2	207.3	121.7	292.9	172.0	412.0	208.0	500.5	237.5	572
	2"	130.8	157.4	185.0	222.5	261.5	314.6	369.4	444.4	519.6	625.1	631.2	759.4	720.8	867
	3"	239.5	192.1	338.7	271.6	478.8	384.0	676.3	542.4	951.4	763.1	1155.7	927.0	1320.	1059
	4"	353.9	216.5	508.9	306.1	719.5	432.8	1016.2	611.3	1429.7	860.1	1736.6	1045.	1983.	1193
	6"	622.8	249.8	880.6	353.2	1245.0	499.3	1758.5	705.2	2474.0	992.2	3005.1	1205	3431	1376
	8"	897.4	269.9	1268.7	381.6	1793.8	539.5	2533.6	762.1	3564.4	1072.1	4329.7	1302	4944	1487
12"	1"	64.0	110.0	96.9	155.5	137.1	219.9	193.6	310.6	272.4	436.9	331	531.	378	606
	2"	215.2	172.6	304.2	244.0	430.1	345.0	607.5	487.2	852.6	683.9	1038	833	1185	951.
	4"	622.8	249.8	880.6	353.2	1245.0	499.3	1758.5	705.2	2474.0	992.2	3005	1205.	3432	1376.
	6"	1102.9	294.9	1559.4	416.9	2204.7	589.4	3114.0	832.6	4381.0	1171.3	5322	1423.	6077	1625.
	8"	1893.2	337.5	2682.9	478.2	3793.1	676.1	5345.3	952.8	7520.1	1340.4	9135	1628	10431	1859
	12"	2730.6	365.0	3860.7	516.1	5458.3	729.7	7709.6	1030.6	10846.4	1450.0	13175	1761	15044.	2011
16"	1"	94.3	113.2	133.0	160.1	188.1	226.3	265.7	319.6	374.	450	454	546	518	624
	2"	298.8	179.7	427.5	254.1	597.3	359.3	843.6	507.5	1187	714	1442	867	1646	990
	4"	897.4	269.9	1268.7	381.6	1793.8	539.5	2533.6	762.1	3564	1072	4330.	1302	4944	1487
	8"	2427.2	365.0	3431.8	516.1	4851.9	729.7	6853.0	1030.6	9641.	1450	11711.	1761	13973	2011
	12"	4160.7	417.2	5882.6	589.8	8316.9	833.9	11747.3	1177.8	16527	1657	20075	2013	22924	2298
	16"	5976.8	449.4	8450.3	635.4	11947.2	898.4	16874.8	1268.9	23741	1785	28837	2168	32929	2476
32"	1"	197.3	118.7	279.0	167.8	394	237	557	335	784	471	952	573	1087.	654
	2"	657.1	197.7	929.1	279.4	1314	395	1855	558	2610	785	3171	954	3621.	1089
	4"	2052.4	308.7	2901.8	436.4	4112	618	5795	872	8153	1226	9903	1489	11308	1701
	8"	5976.8	449.4	8450.3	635.4	11947	898	16875	1269	23741	1785	28837	2168	32979	2476
	16"	15888.8	597.4	22464.3	844.6	31760	1194	44860	1687	63112	2373	76661	2882	87539	3291
	24"	27000.9	676.8	38175.1	956.9	53973	1353	76234	1911	107250	2688	130275	3265	148761	3729
	32"	38659.4	726.8	54658.4	1027.5	77277	1453	109150	2052	153559	2887	186526	3507	212994	4004.

Determination of Width and Depth.—Table 2 gives the volume and velocity of flow of water in rectangular launders of various widths from 4 to 32 in., on various slopes from ⅛ to 4 in. per foot, and for depths of flow from ½ to 32 in. In the computation of this table, the Kutter formula for the flow of water was used, but the values approximate those of the flow of pulp and are safely applicable to launder work.

It is desirable that the depth of flowing pulp be about one-fifth the inside depth of the launder. However, in transmission of frothy pulp, it is necessary that the launder be of greater width in order to increase area. It is recommended that in such cases the minimum width be 10 in., to carry, say, 20 gal. per minute, up to 18 in. wide to carry approximately 250 gal. per minute. Launder sides in both cases to be 12 in. high.

Table 2 is computed within the range of possible requirements for ordinary mill practice. Figures between those given may be found by interpolation or by plotting.

For those who desire to make an original calculation, Kutter's formula is as follows:

$$V = \left\{ \frac{\frac{1.811}{n} + 41.6 + \frac{0.00281}{s}}{1 + \left(41.6 + \frac{0.00281}{s}\right)\frac{n}{\sqrt{r}}} \right\} \times \sqrt{rs}$$

in which V indicates mean velocity in feet per second;

r, hydraulic mean radius, which is the quotient in feet obtained by dividing the area of wet cross section in square feet by the wetted perimeter in feet;

s, the natural sine of the slope angle;

n, the coefficient of rugosity, depending on the nature of the lining or surface of the channel; in this case $n = 0.012$, as for unplaned lumber, perfectly continuous on the inside.

Example: Consider the pulp mentioned in the previous examples. It has been found that the slope required for this pulp is 1 in. per foot. Say 1230 tons of dry ore is milled per 24 hr.; and that the pulp flows at a consistency of 45 per cent solids (by weight). The specific gravity of the dry ore is 2.70. By computation, it is found that the quantity of pulp is 326 gal. per min. Table 2 shows that on a grade of 1-in. per foot this pulp will flow about 2 in. deep in a launder 8 in. wide.

Consider, then, the construction of a launder on this basis. If a 3/8-in. rubber liner is used, the bottom board of the launder may be a 2 by 10, and the side boards 1¼ by 12.

Pulp Formulas

Q, gallons pulp per minute,
p, specific gravity pulp,
a, tons dry ore per day (24 hr.),
b, specific gravity dry ore,
c, per cent solids in pulp,
d, tons pulp per day (24 hr.),
f, tons water in pulp per day (24 hr.),
g, gallons water per minute in pulp,
h, pounds dry ore per minute,
j, gallons dry ore per minute.

Given	Sought	Formulas	
a, c	d	$d = \frac{100a}{c}$	[1]
a, c	f	$f = \frac{100a}{c} - a$	[2]
a, c	g	$g = \frac{\frac{100a}{c} - a}{6}$	[3]
a	h	$h = 1.3889a$	[4]
a, b	j	$j = \frac{a}{6b}$	[5]

(*Pulp Formulas Continued*)

Given	Sought	Formulas	
a, b, c	Q	$Q = \dfrac{\frac{a}{b} + \frac{100a}{c} - a}{6}$	[6]
b, c	p	$p = \dfrac{100}{100 - \frac{c(b-1)}{b}}$	[7]
Q, b, c	a	$a = \dfrac{6Qbc}{c + 100b - bc}$	[8]
p, b	c	$c = \dfrac{100pb - 100b}{pb - p}$	[9]
p, c	b	$b = \dfrac{pc}{100 + pc - 100p}$	[10]

Curved Flumes

Construction of curved wooden launders is, of course, impracticable; but where a flume is to be built of concrete to carry tailings or water, it may be found economical and otherwise desirable to make the necessary changes in direction by means of curves instead of building drop boxes at tangent intersections.

In addition to the determinations relative to slope (or grade) and size of flume—all of which may be made by means of the foregoing text—two others will be required; viz., (1) the amount of "bank" for curves of different radii and (2) the additional slope per foot necessary to compensate for curvature. Formulas for computation of these factors follow:

$$S = \frac{\left(\frac{V}{60}\right)^2}{2.68R} \qquad [11]$$

$$G = S(\sqrt{1 + R^2} - R) \qquad [12]$$

in which V indicates velocity, feet per minute,

R, radius of curve, feet,

S, slope of bank, inches per foot,

G, additional grade required to compensate for curvature.

Example: A concrete flume 32 in. wide, on a grade of $\frac{1}{8}$ in. per foot, will carry 16,000 gal. of water per minute. If the flume is curved on a 40-ft. radius, what banking of the bottom is required? And what additional grade must be provided around the curve? Table 2 shows that the velocity will be 597.4 ft. per minute.

Application of the known values to equation 11 gives:

$$\text{Slope of bank} = \frac{\left(\frac{597.4}{60}\right)^2}{2.68 \times 40} = \frac{99.2}{107.2} = 0.925 = \frac{15}{16} \text{ in. per ft.}$$

Application of known values to equation 12 gives:

Additional grade required to compensate for curvature =

$$0.925(\sqrt{1 + 40^2} - 40) = 0.925 \times 0.0125 = 0.0116 \text{ in.}$$

This figure is so small as to be negligible, being but 1⅛ in. in 100 feet. Grade compensation is required, particularly when the flume carries pulp (e.g., tailings) and reduction in launder grade would not be safe.

Junction Boxes

When it becomes necessary to make a change in direction in a launder line, a junction box is placed at the angle point. Such a box ought to be wide enough, long enough, deep enough and high enough for the purpose, but not too wide, too long, too deep or too high. Safe dimensions of junction boxes may be found through the use of Tables 3 to 7.

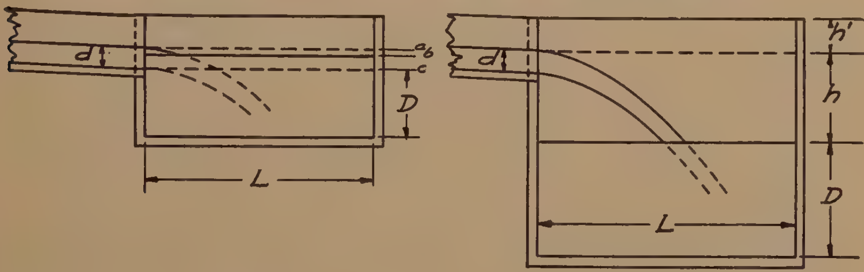

Fig. 3.—Junction boxes with inflowing launders.

Width of Junction Boxes.—Fig. 3 shows junction boxes with inflowing launders. Proper widths of boxes are listed in Table 3.

Table 3.—*Widths of Junction Boxes*

Depth of Inflowing Stream d (see Fig. 3), In.	h (see Fig. 3)							
	½	1	2	4	8	16	32	64
	Values to be Added to Inside Width of Inflowing Launder to Obtain Width of Junction Box							
½	9	9.5	9.5	10	10.5	12	15	20
1	9	9.5	9.5	10.5	10.5	12.5	15.5	20
2	9	9.5	10	10.5	11	13	16	20.5
3	10	10	10	11	11.5	13.5	17	21
4	10	10	10.5	11	12	14	17.5	22.5
6	10	10.5	11	12	13	15	19	23.5
8	11	11	11.5	12.5	13.5	16.5	20.5	25
9	11	11.5	12	13	14	17	21	25.5
12	11	12	12.5	13.5	14.5	18.5	23	27.5
16	12	12.5	13	14	16	20	25.5	30.5
24	12	13	13.5	15	18.5	24	30	36
32	12	13	14	16	21	28	36	42

Given the inside width of the inflowing launder, depth d of the inflowing stream, and height h. From Table 3, find stream depth d at left and from column wherein h is the difference in pulp level (see Fig. 3) take the number of inches. This added to the inside width of the inflowing launder will give the inside width of the junction box.

TABLE 4.—*Lengths of Junction Boxes*

Velocity, V, Ft. per Min.	Depth of Inflowing Stream d (see Fig. 3), In.	h (see Fig. 3),[a] In.							
		½	1	2	4	8	16	32	64
		Length of Junction Boxes L, in. (see Fig. 3)							
60	½	18	18	18	18	19	20	23	24
	2			18	18	19	20	23	24
250	½	18	18	18	18	20	24	31	42
	2			18	19	22	26	33	44
	4				21	23	27	34	45
	8					26	30	37	48
500	½	26	27	28	31	35	42	52	68
	2			31	33	37	44	54	70
	4				35	39	45	55	71
	8					43	49	59	75
	12					46	52	62	78
	16						55	65	81
	24						59	69	85
	32							75	91
750	2			51	57	63	72	85	
	4				59	65	74	87	
	8					69	77	90	
	12					72	81	93	
	16						84	97	
	24						90	102	
	32							108	
1000	2			80	84	89	96		
	4				86	91	98		
	8					93	102		
	12					98	105		
	16						108		
	24						115	132	
	32							144	

[a] h must not be considered to be less than d; e.g., if the pulp level in a junction box is designed to take elevation of line b, Fig. 3a, design the box L as though the pulp level were at line c.

Length of Junction Boxes.—The length of junction boxes can be determined from Table 4 if the velocity and depth of pulp in the inflowing launder and h are known (see Fig. 3).

Depth of Junction Boxes.—The depth of pulp in a junction box must be sufficient to provide ample cushion for the inflowing pulp, thus preventing wear of the bottom of the box. The depth (D) of the junction boxes (Fig. 3) may be determined from Table 5 when the depth d of the inflowing stream, its velocity and h are known.

TABLE 5.—*Depths of Junction Boxes*

Velocity, V, Ft. per Min.	Depth of Inflowing Stream d (see Fig. 3), In.	h, In.							
		½	1	2	4	8	16	32	64
		Depth of Junction Boxes D, In. (see Fig. 3)							
60	½	6	7	8	9	10	11	12	13
	2			8	9	10	11	12	13
250	½	7	7	8	9	10	12	14	16
	2			9	10	11	14	16	18
	4				12	13	16	18	20
	8					15	18	20	22
500	½	8	9	10	11	13	17	21	24
	2			11	12	14	18	22	25
	4				13	15	19	23	27
	8					16	21	25	29
	12					18	22	27	30
	16						23	28	31
	24						24	29	32
	32							30	33
750	2			16	19	22	26	30	
	4				20	23	27	31	
	8					24	28	33	
	12					25	29	34	
	16						30	36	
	24						32	39	
	32							42	
1000	2			23	26	30	36		
	4				27	31	37		
	8					32	39		
	12					33	40		
	16						41		
	24						44	53	
	32							56	

The height (h') of the top of a junction box is usually made flush with the top edge of the inflowing launder. If, as in Fig. 3a, the pulp level in the box is relatively high and the velocity of the inflowing stream great, it may be necessary to add a board at the top of the box, to avoid splash. Therefore, it is well to leave the upper edge of the box boards plowed, to receive an additional board and a cover if they are found necessary.

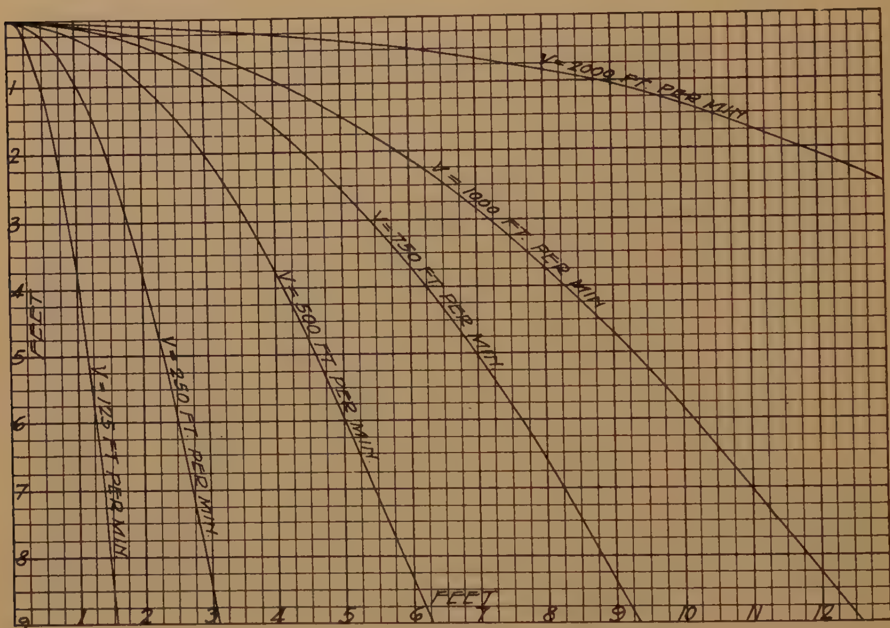

FIG. 4.—PATHS OF DISCHARGING STREAMS.

PULP DISCHARGE

Fig. 4 shows the curved paths of streams discharging at different velocities from the end of a launder.

The head end of the launder, into which pulp is discharged from the junction box, requires special treatment. As the pulp flowing out of the box has no initial velocity, it is obvious that the discharge opening in the junction box must be considerably larger than the intake opening.

The quantity of pulp flowing from a junction box into its discharge launder, the bottom of the launder being flush with the bottom of the discharge opening, is approximately 85 per cent of the discharge over a weir of the same width and at the same depth of flow. Table 6 gives the volume of pulp flowing from the junction box to a launder built as described, "head in inches" being depth of pulp at head of outgoing launder (Fig. 5b).

The size of the discharge launder from the junction box is the same as that of the inflowing launder. It is possible that, owing to lack of head in the box, it may be necessary to widen the discharge opening; which will increase the width of the head end of the discharge launder. If this is done, the inside width of launder is made 6 in. wider than the discharge opening in the box. Each side is tapered thence at the rate of 3:12 until it meets the side of the standard launder section.

TABLE 6.—*Volume of Pulp Flowing from Junction Box to Launder*

Head, In., d'	Length of Opening in Junction Box, In.					
	6	12	18	24	36	60
	Gallons per Minute					
½	5.4	10.8	16.1	21.8	32.7	54.6
1	14.9	30.4	45.9	61.3	92.3	154.1
1½	23.9	51.9	80.0	108.1	164.2	277
2	40.3	83.6	126.8	170	256	429
2½	55.2	115.5	175.8	236	356	598
3	71.5	150.9	230	310	468	786
3½	88.5	188.7	289	389	589	990
4	105.9	228	350	473	717	1206
4½	124.0	270	416	562	853	1437
5	142.5	314	485	656	998	1682
5½	160.8	358	555	751	1145	1933
6	179.7	404	629	853	1303	2201
6½	198.5	452	705	959	1466	2480
7	217	500	782	1065	1631	2762
7½	235	549	863	1177	1805	3060
8	254	600	944	1292	1984	3368
8½	271	650	1028	1406	2164	3677
9	289	701	1114	1526	2352	4002
9½	306	754	1201	1649	2544	4335
10	322	805	1289	1772	2738	4671

Table 6 was computed by the Francis formula:

$$Q = 3.33(L - 0.2H)H^{3/2}$$

in which Q indicates cubic feet per second,
L, length of weir, feet,
H, head, feet.

But

$$\text{Gallons per minute} = 3.33(L - 0.2H)H^{3/2} \times 60 \times 7.48$$

and 85 per cent of this $= 1270.33(L - 0.2H)H^{3/2}$.

Distribution of Pulp

In the distribution of pulp there are two essentials to be considered: (1) that as the volume of pulp fluctuates there shall be a proportionate fluctuation to the several channels into which the main flow is diverted; (2) that the division of pulp shall be effected positively and in a manner that eliminates the "personal equation."

Distribution of pulp in concentrating mills is accomplished in many ways. Methods *A* and *B* described below accomplish the results stated above.

Method A. Weirs.—As shown in Fig. 5, feed enters the box near the top, passes under the baffle and discharges into two launders. If it

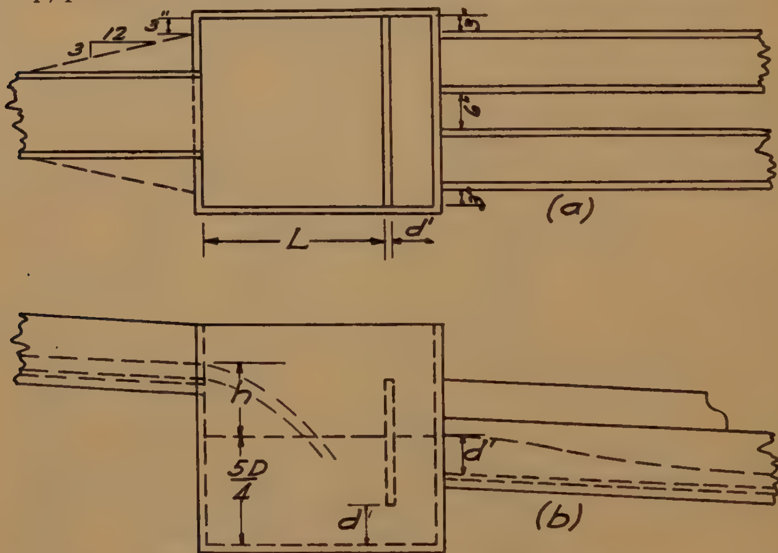

Fig. 5.—Weirs for distribution of pulp.

is desired to split three ways instead of two, the feed launder ought to be flared out to within 6 in. of the full width of the box, as shown by dotted lines (Fig. 5*a*). The length *L* of the box may be determined from Table 4.

The depth of pulp in the distribution box should be 1¼ times the value of *D* given in Table 5. The depth of flow *d'* into the outgoing launders can be found from Table 6. The additional length of box = *d'*, but not less than 5 in. (Fig. 5*a*); also the aperture below the baffle = *d'*, but not less than 6 in. The width of the box is usually governed by the space required for the two or more outgoing launders, but should not be less than the amounts given in Table 3.

The top of the baffle should be high enough to prevent feed from slopping into the outgoing launders but low enough to permit pulp to flow over it if there should be a choke-up under the baffle.

Method B. Bushings.—In Fig. 6 the feed enters the distribution box through the main launder, which is flared at its junction with the box. The pulp passes under the baffle and through porcelain or white-iron bushings into distributing launders.

The dimensions of the box are determined in the following manner: Length L is that given in Table 4; the depth of pulp in the junction box should be not less than $1\frac{1}{4}$ times the value of D as given in Table 5; the additional length of box, X, also the aperture below the baffle, are shown by d' in the left-hand column of Table 6. The baffle should be

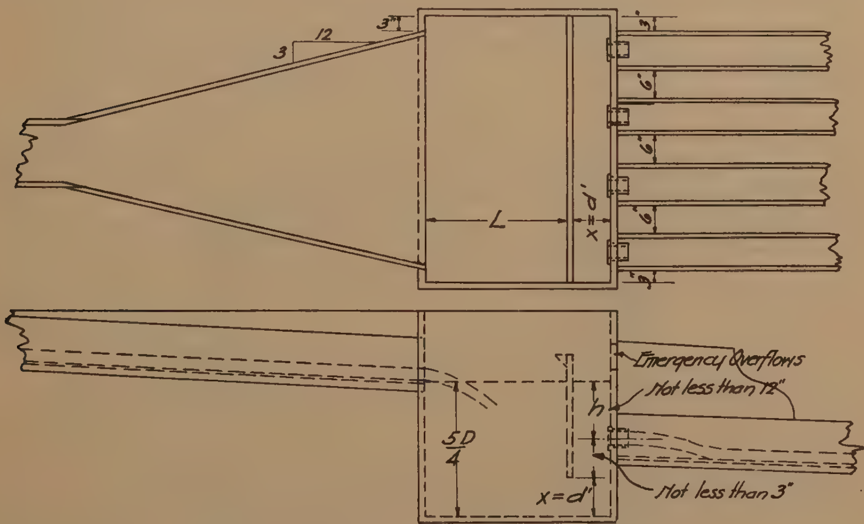

Fig. 6.—Bushings for distribution of pulp.

made as for the box in Fig. 5. It is desirable that the head h be no less than 12 in. In the case of choke-up, it is advisable to provide emergency overflow weirs or holes above the bushings, as shown in Fig. 6.

Discharge of bushings under different heads is given in Table 7, which was computed from the following formula, in which

h indicates head from center of bushing to surface of pulp, ft.,
g, acceleration due to gravity, = 32.2 ft.
V, velocity of freely falling body, ft. per sec. = $\sqrt{2gh}$
Q, gal. per minute.
d, diameter of bushing, in.
C, 0.82; constant for short-tube orifice.

$$Q = \frac{C\sqrt{2gh} \times 12 \times 60 \times (0.7854d^2)}{231}$$

$$Q = 16.1d^2\sqrt{h}.$$

Table 7.—*Discharge of Bushings*

Head, h	Diameter of Bushing, In.													
	¼	½	¾	1	1½	2	2½	3	4	5	6	7	8	9
	U. S. Gallons per Minute													
Inches														
3	0.5	2.0	4.5	8.0	18.1	32.2	50.3	72.5	129	201	290	395	515	652
6	0.7	2.8	6.4	11.4	25.6	45.5	71.1	102	182	285	409	558	729	922
9	0.9	3.5	7.8	13.9	31.4	55.8	87.1	125	223	349	502	683	892	1,129
Feet														
1	1.0	4.0	9.1	16.1	36.2	64.4	101	145	258	403	580	789	1,031	1,304
2	1.4	5.7	12.8	22.8	51.2	91.1	142	205	364	569	820	1,116	1,457	1,844
3	1.7	7.0	15.7	27.9	62.7	111.0	174	251	446	697	1,003	1,366	1,785	2,259
4	2.0	8.1	18.1	32.2	72.5	128	201	290	515	805	1,159	1,578	2,061	2,608
5	2.2	9.0	20.2	36.0	81.0	144	225	324	576	900	1,296	1,764	2,304	2,916
6	2.5	9.9	22.2	39.4	88.7	158	247	355	631	985	1,420	1,932	2,524	3,194
7	2.7	10.7	24.0	42.7	95.8	170	266	383	682	1,065	1,533	2,087	2,726	3,450

Down Spouts

At times it becomes necessary to conduct a pulp to a considerably lower level. This may be accomplished to a certain extent by means of

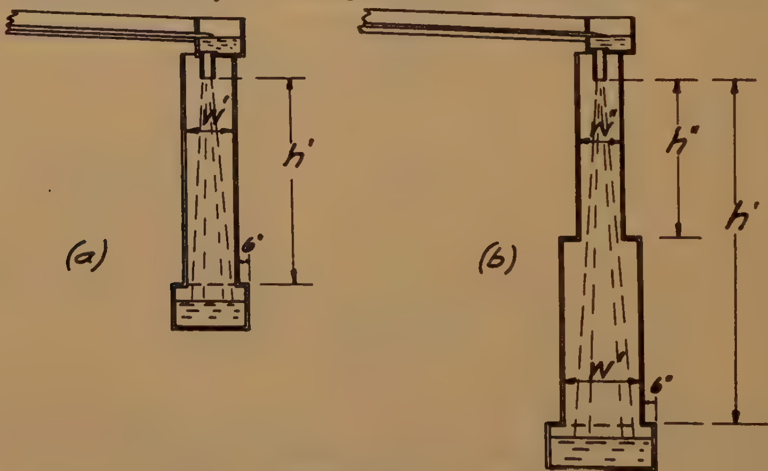

Fig. 7.—Construction of down spouts.

the junction-box arrangement shown in Fig. 3*b*, but when h is much over 16 in. and the pulp velocity exceeds 350 ft. per minute, it would be necessary to build the junction box unbecomingly large. In order to overcome this point, it is desirable to build as shown in Fig. 7*a*, using a junction box in which the pulp level is maintained even with the bottom of the inflowing launder through the use of a white-iron bushing of correct diameter (Table 7). The bushing should be so placed that the inflowing feed does not strike and wear it.

In Fig. 7, the width W', in inches, of the wooden down spout is the inside diameter of the bushing, in inches, plus $2h'$, in feet. When h' is more than 16 ft. it is preferable to step down the upper half of the down spout to the width required at half the depth as shown in Fig. 7*b*.

DISCUSSION

(*C. H. Benedict presiding*)

C. E. Locke,* Cambridge, Mass.—Is there not another factor that should be taken into account; namely, the factor of pulp density, which might involve also the viscosity of the pulp. In the so-called heavy suspension methods of concentration a medium consisting of solid particles in suspension in water is used to separate heavy minerals from light minerals on the basis of specific gravity. Some pulps in launders might partake of the nature of heavy suspensions and thus hold up coarser particles and prevent them from settling out or sanding out in the bottom of the launder. The formation of such suspensions would be affected by the size distribution of the solids in the material being transported. If there existed a considerable proportion of fine particles in the pulp, these fine particles would bring about the suspension effect. If fine particles were absent, there would not exist the suspension effect to hold up the coarser particles.

F. C. Bond,* Milwaukee, Wis.—The method used in Mr. Linke's paper for calculating the suitable launder slope consists: (1) of determining the proper slope necessary to carry the largest particles present, and (2) of determining the proper slope for the existing pulp dilution and specific gravity. The largest slope found is the one selected.

Since variations in the pulp dilution probably result in corresponding variations in the size of the largest particle carried at any specified slope, it would appear preferable to combine the quantities concerned in a single chart. If sufficient data were available, it should be possible to erect a nomograph chart combining particle size, pulp dilution, and the equivalent launder slope, for ore of any specified specific gravity.

R. T. Hancock, London, England.—It is evident that the author has taken a number of observations of launder performances in the Utah Consolidated mill, converted weights to volumes, made a logarithmic plot of these against slopes, derived the corresponding equation from the "best straight line," converted the volumes calculated from this equation to weights again, as more intelligible to the average millman, and from these weight-slope relations drawn the graphs presented in his paper. This is exactly what a qualified investigator would do. The following equations yield his plotted curves with uncanny accuracy:

Porphyry copper ore, sp. gr. 2.70:

$$\text{Slope, in. per ft.} = 11.27\ (\text{per cent volume of solids})^{1.64}$$

Copper concentrates, sp. gr. 3.81:

$$\text{Slope, in. per ft.} = 15.00\ (\text{per cent volume of solids})^{1.185}$$

But such relations are contrary to all human experience; they imply that capacity increases less rapidly than does slope. I suggest that somewhere in the above complicated process the author got his ordinates and abscissas mixed.

H. A. Linke (author's reply).—It is true that if fine particles were absent there would not exist the suspension effect to hold up the coarser particles.

In this mill, where the experimental work was done, the coarsest and driest material handled is drag classifier sand, which carries about 17 per cent minus 100-mesh. This finer material furnishes a certain fluidity, but it is a part of that mill product on which experimental work was done and is one of the products which we are obliged to transport in launders.

Until classifiers are perfected to the point where no minus 100-mesh particles are returned to the ball mills, we shall have to work with drag sands that are somewhat more mobile than clean sand of uniform size particles. At that time, doubtless, it will be necessary to conduct further experiments.

It is possible that a nomograph chart could be constructed and I should be very much interested in Mr. Bond's revision and improvement of my graphs. As they now stand they have served our purpose here for many years; the method is simple and quick, and although the resolution of all into a nomograph would have a certain academic value, I doubt whether the practical value of further expensive research would be justifiable.

* Professor of Mining and Ore Dressing, Massachusetts Institute of Technology.

* Allis-Chalmers Manufacturing Company.

INDEX

(NOTE: In this index the names of authors of papers and discussions and of men referred to are printed in SMALL CAPITALS, and the titles of the papers in *italics*.)

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